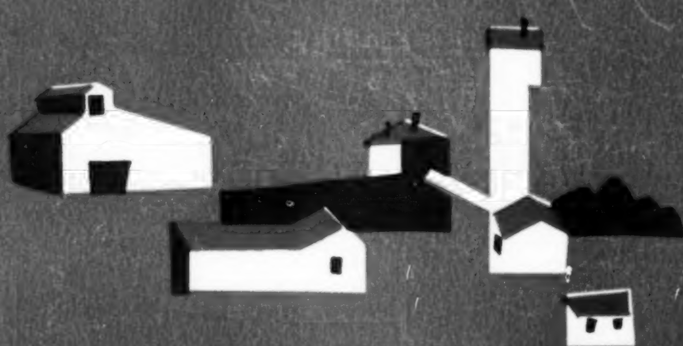
MINING engineering

SEPTEMBER 1953



MILL DESIGN

MINING engineering

VOL. 5 NO. 9 SEPTEMBER, 1953

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COVER

This month Herb McClure's Cover introduces the Mill Design Symposium, starting on page 868.

FEATURES

ARTICLES

TRANSACTIONS

— Personnel Service —

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

MEN AVAILABLE

Executive, 39, bilingual (English-Spanish), U. S. citizen, mechanical engineering graduate, married, no children; 17 years experience mining, mining exploration Latin America, last 7 years asst. manager doing general management, administration, engineering, specializing bilingual executive work such as mining property contract negotiations, government dealings, public relations, etc. Prefers Mexico, Caribbean area, Latin America. M-35.

Mining Engineer, 34, single, M.S. degree, registered P.E.; 13 years experience operation and management in hardrock and placer, including 4 years mine examination, western U. S., Alaska, Canada. Will accept foreign position. Available short notice. M-36-4710-E-5-San Francisco.

Mining Engineer, 26, single, British (Canadian resident), graduated in Mining Engineering, B.Sc. A.R. S.M. Four years all-round underground mining experience. Requires operating or engineering position. Willing to travel anywhere. Location preferred, N. America, Europe or Africa. M-37.

Geologist, Mining Engineer, 26, married, no children, recent graduate B.S. Geology, with good background knowledge of chemistry and mining engineering. Two years experience surveying and heavy construction. Desires position exploration, beneficiation in metallics. Will accept foreign position without single status requirements. Available immediately. Prefers western U. S. or foreign. M-38-536-E-4-San Francisco.

Mining Engineer with record of accomplishment organizing and directing efficient low cost mining and ore dressing operations available for responsible position in United States. Ability based on broad experience with production and engineering, mechanized mining, flotation, safety, union labor relations, and preparation of reports. Married. Employed; available reasonable notice. M-39.

POSITIONS OPEN

Assistant Mine Master Mechanic, 30 to 40, experienced in the maintenance of mine hoists, compressors, ventilating fans, pumps, etc., as well as operation of the mine shops. Salary to start, $6600 a year U. S. cur. Y8989(a).

Diamond Drill Foreman for large diamond drill program, exploring extremity of ore body, grade determination. Salary open. Y8989 (b).

Mining Engineer, young, graduate, with some experience in surveying and mapping. Should be able to assume responsibility and get along well with associates. Duties will include mine and surface surveying and mapping, calculating and performing other related engineering work. Salary to start, $4200 a year; advancement and salary increases. Location, South. Y8371.

Mining Engineers with minumum of ten years experience in maintenance and construction for mining operations; also administrative experience. Location, West. Y8941.

Examiners and Appraisers of independent properties for tin, gold, and diamonds in Belgian Congo and French West Africa. Salary open. Y8932.

Technical Sales Engineer for export department, to assist in technical sales, one who is familiar with overseas operation and able to speak French and German. Should have mining engineering background and be familiar with heavy underground and construction machinery. Salary open. Y8342(d).

Assistant or Associate Professor, M.S. in Mining, 30 or over, to teach general mining engineering courses. Must be interested in making a career of teaching and must have had a least five years experience in metal mining. Good opportunity for advancement. Salary open. Location, Michigan. C1118(a).

Instructor, mining engineer, preferably with Master's degree, to teach general engineering subjects, i.e., drawing; load will be predominantly general in character. Should have both educational and industrial experience, particularly the latter. Salary, $4200 for nine months. Location, South. Y8892(b).

Resident Engineer, graduate, with several years experience in coal mining, including underground surveying and mapping, and development planning, for large western U. S. coal mine. Salary open. Location, West. Y8646(a).

A Progress Report on the Status of FluoSolids

Applications of Non-Catalytic Fluid Techniques Expanding into Many New Fields

Pioneered by Standard Oil Development Company during World War II, fluidization is today widely recognized as an ideal means of promoting intimate contact between solids and gases. As licensee in all non-catalytic non-hydrocarbon fields, The Dorr Company has continually expanded the applications of this new technique since 1944. At the present time, FluoSolids is demonstrating outstanding advantages in the processing of a wide variety of metallic and non-metallic minerals, and holds great promise for the future in other fields not yet fully developed.

GOLD

One of the first applications of the Dorrco FluoSolids System was the roasting of arsenopyrite gold ores prior to cyanidation, and several installations have been operating with marked success for two to five years. Latest development for gold roasting is the split compartment Reactor. In this type of unit, preliminary reducing conditions are provided in one half of the Reactor and a complete oxidizing roast in the other, resulting in decreased cyanide consumption and better metallurgy at the mill.

PYRITE

The sulfur shortage has contributed to great activity in the pulp and paper field. Eight FluoSolids Systems in the U. S., Canada and Norway are either in operation or under construction in sulfite pulp mills, for the production of SO_2 for cooking liquor. Eight more are in operation or being installed to produce SO_2 from pyrite, pyrrhotite and low grade sulfur ores at acid plants, which will produce over 1200 tons per day of H_2SO_4 by the contact method. This impressive record results directly from the fact that FluoSolids can deliver a high strength SO_2 gas at lower investment and operating costs than conventional roasters and provides users with an economically feasible and reliable source of SO_2 despite fluctuations in natural sulfur supply.

ZINC

The first commercial FluoSolids installation for the roasting of zinc concentrate went into operation in the summer of 1952. It is producing SO_2 gas for sulfuric acid manufacture and a desulfurized zinc calcine for leaching, prior to electrolytic zinc production. Results to date have demonstrated marked simplicity of control and operation, and two other zinc producers have ordered similar Systems.

COPPER

More recently, the first FluoSolids System to provide a sulfating roast to a copper-zinc concentrate went into operation. In this application, close operating controls make it possible to render the valuable base metals soluble and, at the same time, minimize solubility of the iron. This permits high recoveries of copper and zinc by leaching. When followed by electrolytic precipitation, economic advantages are indicated under many conditions as compared with conventional smelting practice.

NEW DEVELOPMENTS

Studies to explore the utilization of FluoSolids for the beneficiation of low grade iron ore are now being conducted. In this operation, hematite is given a reducing roast to convert it to magnetite for subsequent concentration by wet magnetic means. Higher unit recovery of iron and the production of better concentrate grades indicates improved economics as compared with other beneficiation methods.

Among the other numerous fields currently under investigation are the calcination of alumina, the reburning of precipitated lime sludge and the self roasting of low grade sulfur ores.

If you would like more information on FluoSolids — the most significant advance in roasting technique in the last 30 years — write The Dorr Company, Stamford, Conn. or in Canada, The Dorr Company, 26 St. Clair Avenue East, Toronto 5.

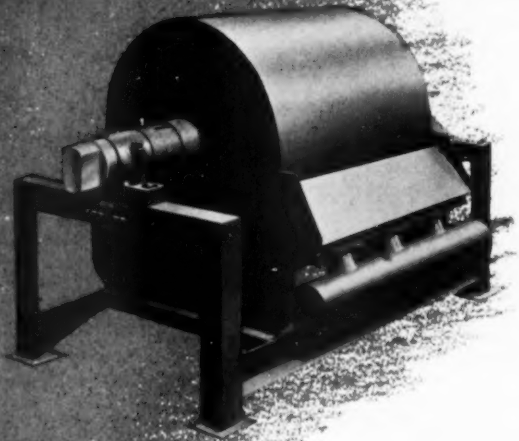
FluoSolids is a trademark of The Dorr Company, Reg. U. S. Pat. Off.

SPEED ORE PRODUCTION

with JEFFREY Crushers . . . Separators . . . Mechanical Vibrating Conveyors

Jeffrey Double Roll 30" x 36" Crusher showing V-Belt drive between rolls.

Jeffrey Type SO Magnetic Separator for heavy media operations.



Jeffrey single section open pan deck MV Conveyor, 18" wide x 15' long.

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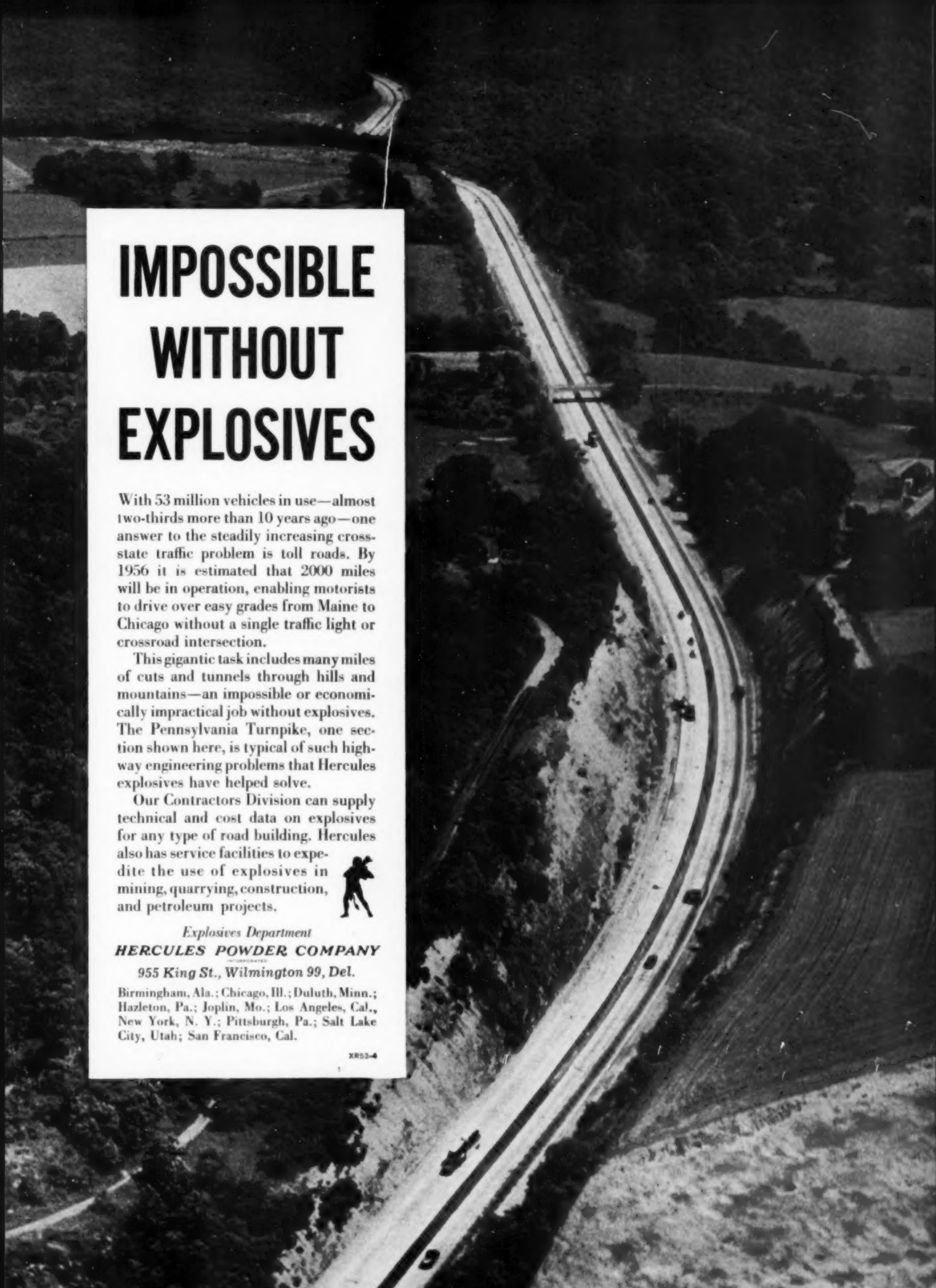

IMPOSSIBLE WITHOUT EXPLOSIVES

With 53 million vehicles in use—almost two-thirds more than 10 years ago—one answer to the steadily increasing cross-state traffic problem is toll roads. By 1956 it is estimated that 2000 miles will be in operation, enabling motorists to drive over easy grades from Maine to Chicago without a single traffic light or crossroad intersection.

This gigantic task includes many miles of cuts and tunnels through hills and mountains—an impossible or economically impractical job without explosives. The Pennsylvania Turnpike, one section shown here, is typical of such highway engineering problems that Hercules explosives have helped solve.

Our Contractors Division can supply technical and cost data on explosives for any type of road building. Hercules also has service facilities to expedite the use of explosives in mining, quarrying, construction, and petroleum projects.

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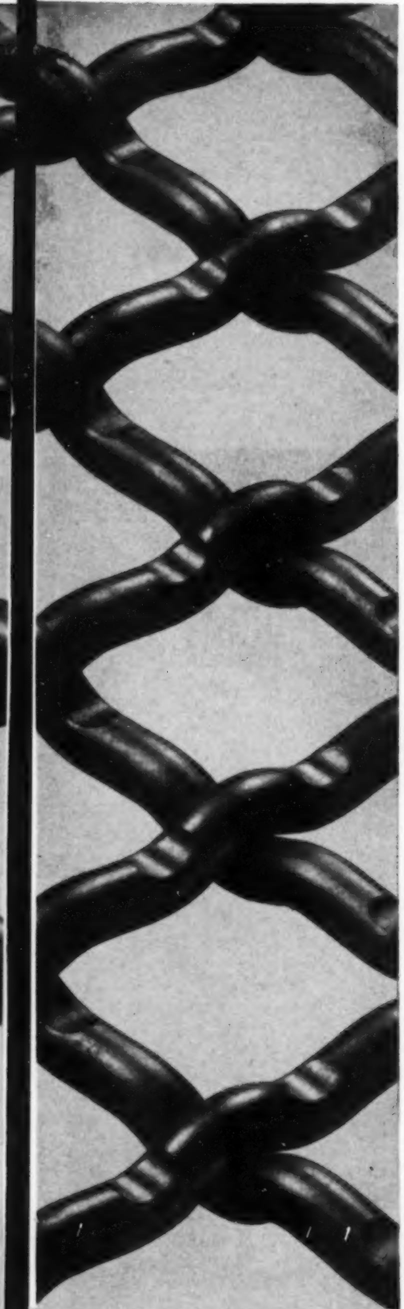
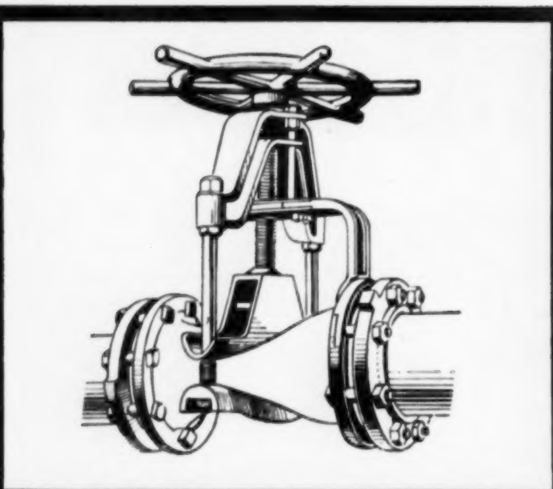
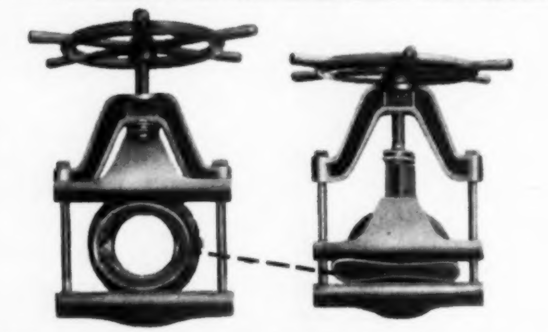




MASSCO-GRIGSBY rubber PINCH VALVES FOR ABRASIVE PULPS



IT'S HINGED...

FOR LONGER WEAR

Recesses molded into opposite sides of the sleeve serve as "hinges" during compression. This patented feature prevents undue strain and eliminates breakage and excessive wear due to valve adjustments.

MOLDED RUBBER RESISTS ABRASION

...gives Long Life without Maintenance

The sleeve is made of molded natural rubber reinforced with fabric; will withstand pressures to 150 lbs. per sq. in., temperatures to 180° F; freezing temperatures are not harmful.

The Massco - Grigsby Rubber Pinch Valve has a straight, unobstructed flow passage which eliminates high friction losses; there are no packing glands; gives positive, leak proof closure; withstands all chemicals not harmful to rubber or neoprene.

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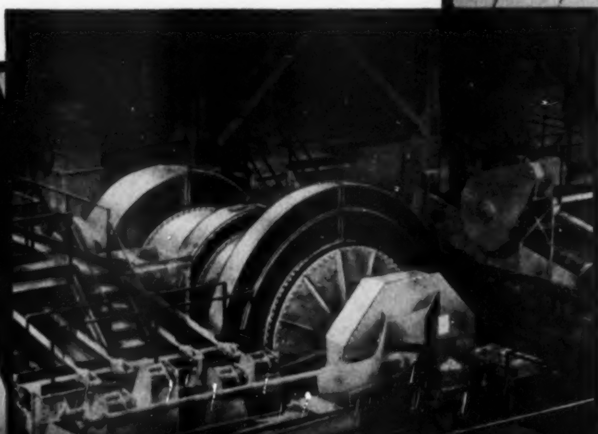

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Books for Engineers

ORDER YOUR BOOKS THROUGH AIME—Address Irene K. Sharp, Book Department. Ten per cent discount given whenever possible. Order Government publications direct from the agency concerned.

Pre-Cambrian Geology of the Picuris Range, north-central New Mexico by Arthur Montgomery, *State Bureau of Mines and Mineral Resources, New Mexico Institute of Mining & Technology;* Bulletin 30, $1.50, 89 pp., 1953.—Petrology and structure of metamorphosed sedimentary and igneous rocks and description of mineral resources. With photographs, tables, two color maps in pocket.

Gas Turbine Analysis and Practice by Burgess H. Jennings and Willard L. Rogers, *McGraw-Hill Book Co.*, $8.50, 487 pp., 1953.—Written primarily for the student, this text is also suited to anyone with a good engineering background. Fundamentals are discussed; then applied to specific components of the gas turbine power plant, with consideration given to thermodynamic design and to stresses and construction materials. Air tables and combustion gas charts are provided.

Old Mill, Homestead, Ore., on the Snake River. From **The Bonanza Trail** by Muriel S. Wolle, *Indiana University Press*, $8.50, 510 pp., 1953. With 108 illustrations by the author. 14 maps. (See *Trends*, p. 863.)

Aluminum in Iron and Steel by Samuel L. Case and Kent R. Van Horn, *John Wiley & Sons Inc.*, $8.50, 478 pp., 1953.—First of a new series of Alloys of Iron Research monographs, this book gives an exhaustive critical review of research on aluminum in modern ferrous metallurgy. Data provided on the effect of small additions of aluminum to molten steel as a deoxidizer and on the effect of aluminum as an alloying element.

Introduction to Solid State Physics by Charles Kittel, *John Wiley & Sons Inc.*, $7.00, 396 pp., 1953.—This introductory text covers crystal structures, thermal and dielectric properties of solids, paramagnetism, ferromagnetism, superconductivity, the free electron and band theories of metals, semiconductors, and imperfections in solids.

Symposium on the Use of Radioisotopes in Soil Mechanics (Special Technical Publication No. 134) *American Society for Testing Materials*, Philadelphia, $1.25, 34 pp., 1953.—Contains three research papers covering methods of measurement of soil moisture and density by radioactive materials and by nuclear radiations. Subject also discussed in general.

Diesel Engine Catalog, Volume 18, Edited and published by *Rex W. Wadman, Diesel Engines Inc.*, Los Angeles, $10.00, 369 pp., 1953.—Standard guide brought up to date. Detailed text descriptions of important engines are illustrated by diagrams and cross-section drawings. Specification tables, classified directory of engine and accessory manufacturers.

Mining Year Book 1953, compiled by Walter E. Skinner, *Walter E. Skinner and The Financial Times,* London, $7.00 post free, 856 pp., May 1953.—This list of 966 world-wide mining companies and mining finance companies includes directors and other officials, incorporation date, description of property, present working results, ore reserves, capital, dividends, prices of shares for the last three years, etc. Also included are the names and addresses of 1160 mining engineers, managers, etc.; a list of firms supplying mining machinery and equipment, etc.

Pegmatite Deposits of the White Picacho District, Maricopa and Yavapai Counties, Arizona by Richard H. Jahns, *University of Arizona,* $1.25 (Free to residents of Arizona), 105 pp., Nov. 1952.—Arizona Bureau of Mines Mineral Technology Series No. 46 Bulletin No. 162. With photographs, tables, and nine supplementary maps.

The Union Contract Clause Finder, *National Foremen's Institute,* New London, Conn., $7.50, 91 pp., looseleaf, fabricoid binder.—This book is based on an analysis of 3000 union contracts. Detailed discussions of contract problems are accompanied by sample contract clauses which repeatedly show up in management-union negotiations. Current labor legislation is reviewed in the opening section.

Basic Statistics of Industrial Production 1913-1952, *The Organisation for European Economic Co-operation,* 1 vol., $1.50, 110 pp., bilingual French/English. — This book of OEEC statistical bulletins contains data relating to industrial production in Western Europe. Tables include series for OEEC member countries, their overseas territories, and the United States and Canada. In some tables it has been possible to include a column for USSR production.

Handbuch de Metallbeizerei by Otto Vogel, *Verlag Chemie,* Germany, Second Edition 1951. **Volume I: Nichteisenmetalle,** $49.20, 410 pp., **Volume II: Eisenwerkstoffe,** $63.00, 538 pp.—This comprehensive handbook in German covers nonferrous metals in Volume I and ferrous metals in Volume II. Each volume is complete in itself with individual author and subject indexes and classified directory to German sources of supply. The volumes are divided into two parts: a general section covering plant and equipment, materials, waste disposal, and hazards; a practical section covering a wide range of pickling and cleaning processes. The new edition has been thoroughly revised.

Letter to the Editor

A Clarification

On page 152 of the February issue (MINING ENGINEERING) there is the statement, 'Barret reports that the discovery of the Indian Creek orebody in Missouri followed Radore exploration.' On page 345 of the April issue it is pointed out by Dr. Brown that the Indian Creek orebody was discovered in 1947-50 through geological exploration, and the Radore indications of an extension were obtained in 1951, and were subsequently confirmed by drilling. The inaccuracy in the February review occurred in the condensation by the author of material supplied to him by William M. Barret Inc. It should be added that William M. Barret Inc. has never claimed credit for the discovery of the Indian Creek orebody; that the only claims made by Barret in this regard were previously approved by Dr. Brown personally; and that a controversy, such as is implied in the material in the February and April issues, does not exist.

Glenn J. Baker
William M. Barret, Inc.
Shreveport, La.

LOADING IN RECORD TIME

Contractors on high speed jobs and mining companies on regular production or development work prefer Eimco 40H loaders for faster, more dependable loading.

On high speed jobs they prefer the Eimco 40H for maximum loading of large cars and to keep the loading portion of the cycle shorter than drilling time.

On a recent high speed tunnel the time consumed in moving the 5 machine jumbo to the face and drilling was 65 minutes with no time charged for moving the jumbo out during the powder loading operation while time consumed for moving the loader in, loading and moving the loader out was only 55 minutes.

The 40H is a heavy-duty machine for operation under the most severe conditions. Other Eimco loaders are made to fit any loading condition with capacities to 450 yards per minute.

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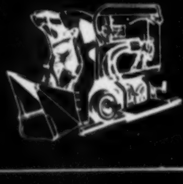

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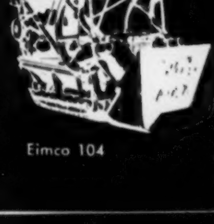

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Eimcos are built heavy, for dependable operation on the roughest, toughest jobs; for handling rock, sticky clay, gravel or any combination of hard to load materials.

With Eimcos you get a faster loading cycle — forward to dig and load back to discharge into the truck or railroad car. This action makes it easier on tracks and rollers for less maintenance and repair.

Quantities loaded can run from 300 to 350 yards per hour depending on your conditions.

Let an Eimco representative show you a machine working near you. Write for more information.

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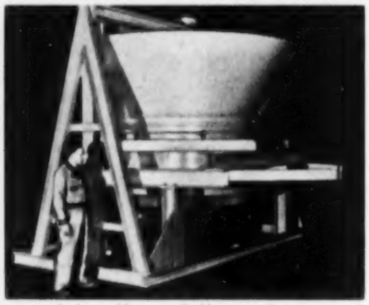
Feeder

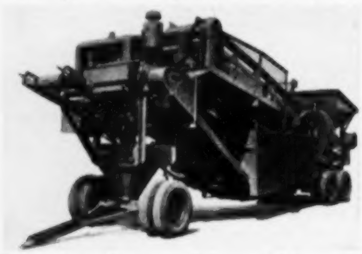
Largest Com-Bin feeder built to date, 11-ft diam, is now in use receiving superphosphate in 10,000-lb charges and delivering 65 tph to a belt conveyor. Smallest model in



use, 8-in. diam, delivers 3 g per 8 sec. Manufactured by *Pulva Corp.*, the feeder is capable of handling sticky material. Cylindrical bin and table both rotate as a stationary plow removes stream of material. **Circle No. 1**

Crushing Plant

Pioneer Engineering Works Inc. has added a duplex crushing and screening plant to its line of portable plants. Equipped with the



largest units possible, which still comply with weight limitations on road movement, crushers, screens, and feeders are mounted on welded steel truck frame. **Circle No. 2**

Hole-A-Matic

A lightweight, one-man operated, portable hole digger and tunneler developed by *Eugene John Freeman & Co.* digs 4 to 8 in. diam holes up to 6 ft deep. **Circle No. 3**

Assemble-it-yourself

Leebaw Mfg. Co. ships all-steel floor trucks that can be put together "in less than five minutes." No bolts, nuts, screws, etc.; all sections are securely slip-fitted for easy, fast assembly or removal. **Circle No. 4**

Flexible Hose

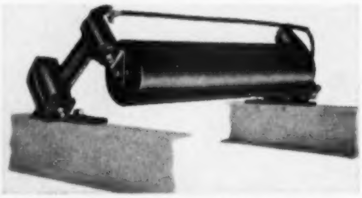
The Flexaust Co. has introduced Portovent, a flexible hose in 4 to 36-in. diam sizes. Cotton or nylon fabric body is Neoprene impregnated and spiral wire reinforced. **Circle No. 5**

Sump Pump

Acidproof 1½ in. sump pump introduced by *The Galigher Co.* handles acid sludges, corrosive and foamy products with solids in suspension at up to 75 gpm flow and 35 ft head. Suitable for abrasive pulp and acid transfer in mills and smelters, floor clean-up, draining and filling acid tanks, the welded steel vertical pump is completely rubber lined and covered. Overhung suspended shaft design eliminates all submerged bearings, packings, and other parts that normally cause trouble. **Circle No. 6**

Conveyor Belt Idler

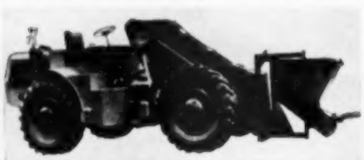
Chain Belt Co. has developed a conveyor belt return training idler, providing automatic alignment for the return belt without the use of



side guide idlers. Shift of belt to either side results in compensating tilt by training idler that guides belt back to position. **Circle No. 7**

Attachments

Five new attachments: lift forks, backfill blade, crane boom, concrete hopper, and dozer blade, increase flexibility of Scoopmobile model LD10 and LD5 front end loaders.



Attachments mount directly on the hydraulic boom and adapt the power steered loaders built by *Mixermobile Mfrs.* for specialized tasks. **Circle No. 8**

Centriclone

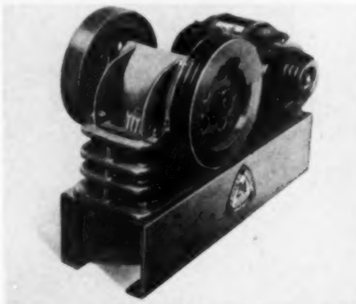
Oliver United Filters Inc. has acquired all manufacturing, sales, and service rights to the Centriclone, unique combination of liquid cyclone and centrifuge, from the former owners, Equipment Engineers Inc. **Circle No. 9**

Loaders

Le Roi Co. has announced formation of a new Le Roi-Transo Div. for manufacture of front end loaders and truck mounted mixers.

Laboratory Crusher

New Denver 3¼x4½ in. forced feed laboratory jaw crusher shown is powered by 2 hp motor and has capacity of 500 lb per hr of —¼ in. material. Crusher takes its place in



Denver Equipment Co.'s complete line of forced feed units ranging from a 2¼x3½ in. laboratory unit to 32x40-in. model. **Circle No. 10**

Coal Hauler

Fleet of seven experimental lightweight coal haulers, built of recently developed A45S aluminum-magnesium alloy, is being delivered to Alcoa's Rockdale, Texas, plant by

Euclid Road Machinery Co. New trailers have 45-ton capacity, 5 tons greater than the standard Euclid coal haulers of equivalent power. Gross weight of tractor and trailer is 8200 lb less than for the 40-ton steel model. **Circle No. 11**

Roof Bolting

Roof-Lock steel expansion anchor for roof bolting developed by *Equipment Corp. of America* features work hardened ribbing for tightening varying diameters. **Circle No. 12**

Color in Hats

Hard Boiled hats from *E. D. Bullard Co.* now come in vivid colors to designate occupation or for safety. Color is impregnated in the fiber glass. Hats that glow in the dark for miners, night crews, and watchmen are also available. The phosphorescent finish is molded into the crown. **Circle No. 13**

Research & Testing

Hunter Associates Laboratory has released a description of its services for research, instrumentation, and technical service on optical appearance of raw materials and manufactured products. **Circle No. 14**

Free Literature

(21) DC MOTORS: *Louis Allis Co.* has released a 16-page bulletin covering a line of dc motors, ½ to 300 hp, and dc generators, ¾ to 250 kw. Also included is a section on motor-generator sets for converting ac to dc in industrial service.

(22) DUMPERS: Covering latest developments in mine car dumping and control equipment, *Nolan Co.'s* catalog pays particular attention to the three models of the new Nolan Porta-Feeder, which have direct mechanical drive, hydraulic fluid drive, and hydraulic cylinder drive.

(23) MEASURING EQUIPMENT: A revised edition of *General Electric Co.'s* 64-page measuring equipment catalog contains information on products ranging from simple current indicators to completely automatic oscillographs; from surface roughness scales to mass spectrometers; from dc amplifiers to radiation monitors. A brief description of each product and its field of application, condensed tables of important characteristics and prices are given.

(24) ELECTRONIC WEIGHING: How to weigh the contents of tanks, bins, and hoppers electronically with Baldwin SR-4 load cells is explained in an 8-page, illustrated brochure from *Baldwin-Lima-Hamilton Corp.* Also covered are installation design considerations, such as mountings, temperature effects, and vibration; indicating and recording instruments; weighing accuracy; and B-L-H's engineering services.

(25) TANTALUM & ZIRCONIUM: *Murex Ltd.* has published a booklet giving data, largely through tables, on tantalum and zirconium. U.S. representative of this English firm is *C. Tennant, Sons & Co.*

(26) INTERCOM SYSTEM: Bulletin from *Farmers Engineering & Mfg. Co.* describes Femco Pagephone which uses one central amplifier, one common cable, requires no operator, and is said to be an inexpensive means of replacing complex PBX switchboards.

(27) PUMP: Bulletin on *Carver's* 1½ in. Lightweight Champ, weighing 55 lb and pumping 100 gpm,

suggests using this pump for auxiliary fire protection, dewatering flooded areas, and draining sumps.

(28) VOLT-AMP TESTER: *Pyramid Instrument Corp.* has information on a pocket size tester for checking voltage and amperage without breaking the circuit. The 11-oz Junior snap-around volt-ammeter costs about $20.00, including voltage test leads.

(29) BENEFICIATION: Types of electrical equipment and systems best suited for each step in mining and processing iron ore from pit operations, through transportation, milling, and concentrating, townsite planning and power generating and distribution are covered in a 20-page booklet from *Westinghouse Electric Corp.*

(30) CRUSHER DATA: Information on how to most efficiently and satisfactorily apply crushers to specific jobs has been published by the *Pennsylvania Crusher Co.* All important factors influencing operation: power consumption, parts wear, maintenance costs, uniformity of product, and others are discussed in detail. A check list helps determine the best type of crusher for various jobs. Hammermills, jaws, impactors, granulators, gyracones, Bradmills, Bradford breakers, Bradford hammermills, and single rolls are discussed.

(31) REGULATOR: *The Askania Unit Regulator Co.* has a self-contained hydraulic controller engineered for automatic pressure, flow and proportioning control. Operating principles of different unit models, application diagrams, and specifications are given in an 8-page bulletin.

(32) WIRE & WISDOM: "Through the Meshes" has been published continuously for 44 years by *W. S. Tyler Co.*, makers of woven wire screen and screening machinery, elevator cars and entrances. This amiable booklet is free to all within the business range of the company. To others the price is 10¢. Among other things, the current issue discusses licorice, Iceland, and the dearth of good pitchers and playwrights.

(33) MECHANICAL LOADING: Bulletin from *The Eimco Corp.* shows cost figures on several operations where chute and grizzly systems have been changed over to drawpoint loading with Eimco loaders. Wider drawpoint spacing and elimination of grizzly level are advantages leading to cost savings.

MAIL THIS CARD

for more information on items described in Manufacturers News and for bulletins and catalogs listed in the Free Literature section.

Mining Engineering **29 West 39th St.** **New York 18, N. Y.**

Not good after December 15, 1953 — if mailed in U. S. or Canada

Please send me { More Information ☐ / Free Literature ☐ / Price Data ☐ } on items circled.

1	2	3	4	5	6	7	8	9	10
11	12	13	14	15	16	17	18	19	20
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51	52	53	54	55	56	57	58	59	60

Students are requested to write direct to the manufacturer.

Name..Title..........

Company..

Street...

City and Zone..State..........

(34) COAL CLEANING: A 6-page, two-color bulletin from *The Dorr Company* entitled "Dorr Equipment and Methods for the Modern Coal Cleaning Plant" covers equipment originally designed to solve basic classification and sedimentation problems in metallurgical and industrial applications. Their uses in the coal cleaning plant are described by photographs, size and capacity ratings, applications, and in typical plant flowsheets.

(35) DIAMOND DRILLS: *Anton Smit & Co Inc.* has published an attractive, well-illustrated, 16-page book. File size and tab indexed, this

catalog contains information on the latest developments in diamond drill bit products.

(36) CLASSIFIER: An illustrated 20-page bulletin from *Western Machinery Co.* presents design features and performance characteristics of the Wemco S-H Classifier units. Also included are operating notes and calculations used in wet classification processing. Handy reference tables list classifier specifications and capacities.

(37) COMBUSTION DATA: *Hauck Manufacturing Co.* has published the 3rd edition of a handbook on industrial heat processing applications. Its 168 pages are directed at those concerned with the maintenance and operation of oil and gas combustion equipment in industrial furnaces, ovens, calciners, etc.

(38) DIRT MOVERS: The tractor div. of *Allis-Chalmers Mfg. Co.* has a 12-page three-color illustrated catalog covering seven modern pull-type scrapers ranging in size from 2 to 18 cu yd struck capacity. Included are complete specifications and other pertinent data.

(39) AIR HOSE: A bulletin from *Quaker Rubber Corp.* describes a number of different uses for air hose, varying from heavy-duty construction through digging and drilling operations to many heavy and light industrial and commercial uses. Performance and specification data are included.

(40) V-TYPE ENGINES: *Nordberg Mfg. Co.'s* 13 in. bore, 16½ in. stroke Supairthermal V-type engines are built to meet demand for a high power, heavy duty, diesel, Duafuel, or spark-fired gas engine of moderate size. These 12 and 16 cylinder engines delivering 2400 to 4260 hp at 450 to 600 rpm are described in an 8-page two-color bulletin.

(41) TRIPLE SUPERPHOSPHATE: Available from *The Dorr Company* is a technical article, "The Manufacture of Triple Superphosphate," which was presented at the Fertiliser Society in London in January. The paper covers various aspects, including the chemical reaction between phosphoric acid and phosphate rock, and general methods of manufacture.

(42) LITHIUM: Guide book on lithium and its compounds, available from *Foote Mineral Co.*, contains many new ideas and new applications of lithium chemicals, as well as complete coverage of current commercial applications.

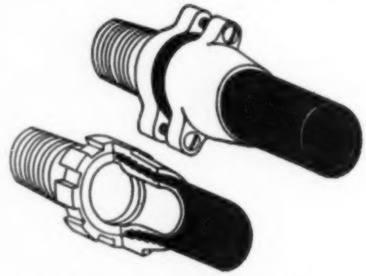
(43) BACK HOE BUCKETS: *American Manganese Steel Div.* has a catalog sheet on Amsco all-manganese steel back hoe buckets, giving features and showing types of side cutters available.

(44) BATTERIES: A brief inspection tour of the engineering research laboratories of *The Electric Storage Battery Co.* and the recently enlarged Crescentville plant at Philadelphia is possible through the pages of an illustrated booklet from the makers of Exide batteries.

(45) ALUMINUM MINE PROPS: Mine supports of aluminum, widely used in German and English coal mines are described in literature available from U.S. representative of *Wiemann-Maschinenfabrik Gmbh.*, the German manufacturer. The new lightweight props are stated to save labor, last longer, and to improve safety. Photographs and text show props used in variety of situations underground.

(46) RECTIFIERS: A brochure from *American Rectifier Corp.* explains applications for standard unit rectifiers, now available from stock for prompt delivery. Units range from 3 to 50 kw, 125 or 230v dc, output and accommodate any ac voltage or frequency input.

(47) NONSLIP PIPE FITTINGS: Barracuda brass fittings, offered by the *Nelson Foundry Co.*, are said to



provide a new and efficient way to connect plastic pipe. A booklet shows how the teeth grip both inside and outside of pipe to overcome cold flow property of plastic.

(48) CONTROL SERVICES: "A Proposal for Managements with Vision," is title of new brochure from *CDC Control Services Inc.* describing a specialized engineering service for consultation, analysis, design, and construction in the automatic control field.

(49) FIRE PROTECTION: The MSA mine fire truck pictured and described in bulletin issued by *Mine Safety Appliances Co.* has 1000-gal water capacity, pump, hose, combination fog and stream nozzle, and space for auxiliary firefighting equipment. Unit is mounted on four-wheel truck with chilled-iron wheels.

(50) FAN ENGINEERING: *Aerovent Fan Co.* offers a 64-page illustrated guide to modern industrial ventilation and air movement. Manual gives engineering installation and operation data for standard and special duty fan applications.

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MINING ENGINEERING

29 WEST 39th STREET

NEW YORK 18, N. Y.

HERE'S WHY

this remarkable Rear-Dump belongs in your present fleet

Hauls anywhere . . . Tournarocker is a new kind of haul unit built from the ground up for "off-road" hauling. It safely travels mountain trails and cross-country as well as highways and city streets. It hauls capacity loads over any kind of terrain. And, it stands up to tough, rough hauling work on steep grades with a minimum of maintenance.

Cuts weather delays . . . Giant tires increase flotation. There are no small idler steering wheels to rob power and traction. Tournamatic differential applies extra power to drive wheel on firmest footing . . . pulls unit through mud, sand, and snow which stop ordinary hauling units.

Reduces maintenance . . . because Tournarocker has no hydraulics, high-pressure jack lines, long drive shafts, frame, sub-frame, springs, or tie rods, the most common troubles of other rear-dumps are eliminated. Large single tires have no divided face to collect rock fragments that wedge in, wear and tear. Three-layer, grid-type bowl with tool-steel floor resists shock of dumped-in rock.

Speeds loading . . . Excavator operator gets faster load cycles because he has a wide top opening for a target. He gets less spillage which saves clean-up work. He swings in and out faster because of wide low entry at rear of body.

Saves spotting time . . . 90° turns and positive power steer quickly position "big-target" body under dipper. No switching back and forth or turntables necessary.

Cleans load every time . . . Even in sticky clay, Tournarocker dumps clean. Streamlined body sheds material readily . . . none rides back to excavator to steal payload room on next trip. Induction body heating available for winter work.

Works fast in tight quarters . . . push-button power steer, 90° turns, electric controls, plus multi-disc air brakes that have more braking surface on a *single* wheel than most hauling units have on *all four*, all contribute to fast, safe haul.

Reduces dumping time . . . a touch of electric switch activates hoist motor. Body lifts smoothly for instant dump. You get "power up" and "power down". There's no delay for hydraulic pressure to build up . . . no shock loads of free gravity dump. Powerful braking action of 4-wheel disc air brakes lets operator back up fast to edge of fill.

Saves dump clean-up . . . front-wheel drive and powerful brakes make it easy for operator to spot safely for dump over edge of bank. Body swings behind and below rear wheels, dropping rocks and dirt completely over the edge, greatly reducing dump clean-up. This clean dumping permits continuous free spotting along entire fill.

Design provides comfort . . . and safety for operator. Big low-pressure tires and air-foam cushion seat smooth out bumps . . . cut the "up-and-down ride" for driver. Push-button controls on instrument panel activate electric motors at point of action. Heavy manual work fighting steering wheel and control levers has been eliminated. Operator dumps, steers, shifts electrically. Fatigue factors are greatly reduced.

Improves safety . . . Low center of gravity, high clearance, good visibility, big brakes, front-wheel drive, giant tires, big top opening and ease of control, all help make Tournarocker the leader in hauling safety.

Insures future earnings . . . Behind the 2-wheel Tournarocker prime-mover, you can interchange a scraper, bottom-dump, crane, or flatbed. Any of these trailing units are available at about 25% of initial cost. With them, you can always fit future operations, insure bigger annual earnings.

Tournarockers available in 9, 18, 35, and 50-ton sizes.

See your LeTourneau-Westinghouse Distributor for all the facts. He'll be glad to show you performance figures from jobs like yours, or to give you names of Tournarocker owners.

Tournarocker—Trademark Reg. U.S. Pat. Off. R-406-G-b

LeTOURNEAU LeT W WESTINGHOUSE

LeTourneau-Westinghouse Company

PEORIA, ILLINOIS

New Jersey Zinc Co. is going ahead with development of a lead-free zinc sulphide orebody one mile south of Jefferson City, Tenn. A 1200-ft shaft will be sunk later this year. Plans call for construction of a 1000-ton per day mill. New Jersey Zinc acquired the property in 1947 and started exploratory drilling in 1948. The zinc company is also contemplating copper exploration on a 77-claim property in the Jerome district of Arizona under a three-year lease from Verde Exploration Ltd.

Special issue Dec. '53 ME

Southern Coal Producers Association reports that 115 mines closed in a four-state area in the last 2½ years. Payroll loss in the region—West Virginia, Kentucky, Tennessee, and Virginia—amounted to $50 million annually. Mines were reported to have employed 15,399 men.

Sylvania Electric Products Co. and Minerals Engineering Co. of Grand Junction, Colo., organized the Salt Lake Tungsten Co. to build a tungsten refinery at Salt Lake City. President of the new company is Blair Burwell, head of Minerals Engineering.

Defense Minerals Exploration Administration has signed 517 contracts under the national defense program to discover and develop critical and strategic mineral resources. Total was reached with the signing of 46 new exploration pacts in the second quarter. Cost of the 517 contracts was put at $25,171,583, with Government's share reaching $15,204,434.

Foote Mineral Co. reports production at Kings Mountain, N.C., facilities has reached a satisfactory rate for lithium concentrates. The lithium chemical plant at Sunbright, Va., is expected to go into production soon. The company is intensifying its search for lithium and other minerals.

- See page 890

U. S. Steel Corp. officials broke ground for a modern research center devoted to new processes for the making of steel. Experimental equipment for study of steelmaking from raw material stages to finished product will be housed in three buildings with a total floor area of 132,000 sq ft. U. S. Steel has laboratories at Pittsburgh and Kearny, N.J., as well as the projected unit at Monroeville, Pa.

Saguenay Terminals Ltd. of Montreal will extend its shipping operations to include a Jamaica-Kitimat Line to supply Aluminum Co. of Canada's Kitimat, B. C., plant with about 160,000 long tons of alumina annually. Saguenay is the shipping arm of Alcan. Carrying alumina instead of bauxite will reduce cargo weight by 50 pct. Alumina Jamaica Ltd., Alcan subsidiary, will refine bauxite at Jamaica.

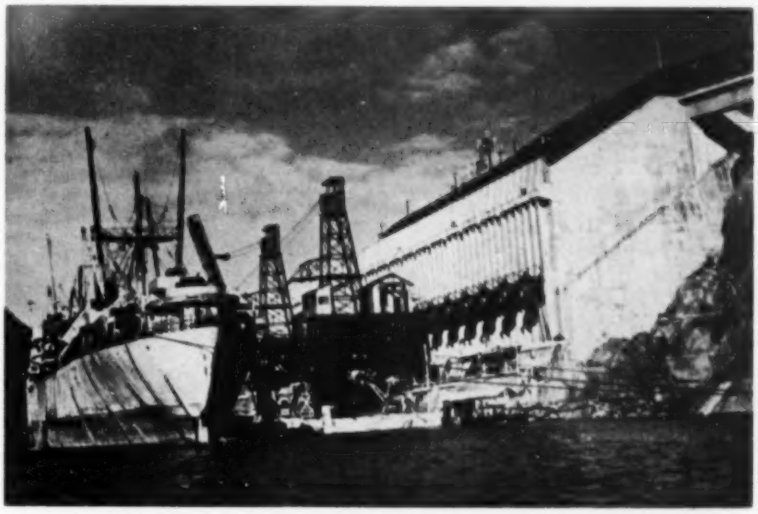
Consolidated Mining & Smelting Co. will spend $1.9 million to string power lines between the firm's Waneta power plant and its Trail, B. C., operations. Three 69-kv lines will carry power about 11 miles when the Waneta plant starts in 1954.

St. Joe Continues Canadian Exploration

Substantial tonnage of commercial lead, zinc, and copper ore with pyrite is indicated by diamond drilling at 78 claims in New Brunswick, Canada. These claims are owned jointly by St. Joseph Lead Co.'s wholly-owned subsidiary, Leadridge Mining Co. Ltd., with Anacon Lead Mines Ltd. and M. J. Boylen et al.

Active exploration continues on these claims and others held by Leadridge in the same area. St. Joe also announced that recent developments in its oil exploration have been encouraging. A second test well, drilled jointly with Continental Oil, was completed on the Harris Ranch block, Crockett County, Texas. A July test indicated a flow of 722.8 barrels of 47 pct gravity oil through a 34/64 in. choke during a 14-hr period, with a strong flow of gas.

Consolidated net income for St. Joe for the six month period ending June 30, 1953 was $4,402,746.26 after provision for taxes on income of $2,355,160.06, according to the company report for the period.



Iron ore being loaded at docks of Cia. Vale do Rio Doce, at port of Vitoria, Brazil. Large ore bunker has series of three Link-Belt traveling loading towers.

Mines Bureau Award To J&L Open Pits

Bureau of Mines is awarding Certificates of Achievement to four of five Jones & Laughlin Steel Corp.'s Minnesota ore div. open pit mines for 1952 operation without a lost-time injury.

Award-winning J&L mines are: the Hill Annex, Longyear, Columbia, and Wentworth mines at Calumet, Hibbing, Virginia, and Aurora, Minn. The J&L mines were among 145 mines, quarries, and open pits in the U. S. that operated 17 million man-hours during 1952 without a lost-time injury.

J&L also operates the Schley mine at Gilbert, Minn., which produces more than 3 million tons of iron ore annually. Two other J&L mines on the Mesabi range are expected to start production in 1955.

Vanadium Corp. Installs Roaster at Naturita

Atomic Energy Commission granted Vanadium Corp. of America permission to install a roaster for processing vanadium-uranium ores at its Naturita, Colo., mill.

Ore comes from Vanadium Corp. of America property and from independent producers. The company exercised its option to purchase the mill at Durango, Colo., which also processes vanadium-uranium ores. The mill has been operated by the company for five years under lease from AEC.

360 Mile Phone Line To Knob Lake Mines

Engineers are constructing a 360-mile telephone and telegraph line between Knob Lake in Labrador and Seven Islands on the Gulf of St. Lawrence. The line will link the new Labrador iron ore region with civilization.

It will also provide traffic control for the single-track railroad between the two places, according to A. K. Hansen, Quebec North Shore & Labrador Railway.

Radio communications were discarded and it was decided to build a separate power line carrier using the same poles that carry 23 kv, single phase balanced power line between the two points, along the railroad right of way.

Nine trunk voice channels will be provided between Seven Islands and Knob Lake. A switchboard at Seven Islands will control all traffic over the trunk channels to Oreway and Knob Lake.

Seven Islands will be the outlet for ore coming from the Hollinger-Hanna mines at Knob Lake. The so-called Ungava mines are said to have a production expectancy of up to 20 million tons annually.

Civil Service Loses Nine Mines Bureau Jobs

Nine key posts in the U.S. Bureau of Mines are no longer under civil service jurisdiction.

Positions are chief mining engineer, chief metallurgist, chief fuels technologist, chief economist, assistant to the director, special assistant to the director, and three assistant directors for programming, health and safety, and mining engineering.

Reserve Builds Town At Beaver Bay, Minn.

Construction is scheduled to start "immediately" on 300 new homes and an 18-room school at Reserve Mining Co.'s new town near Beaver Bay, Minn.

Workers at the huge taconite plant under construction there will live in single story, ranch style houses. There will be ten different exterior combinations with 255 three-bedroom houses and 45 two-bedroom houses in this 1953 group.

Reserve erected 64 housing units at this townsite in 1952. At Babbitt, site of Reserve's taconite mine and small processing plant, 129 houses were built.

Combined population for the two towns is expected to reach 9000 by 1957, when Beaver Bay plant is scheduled to produce 3.75 million tons of iron ore pellets from taconite annually.

Two More Groups Plan Investigation of Minerals

Shortly before the Congressional recess, two more investigations into the minerals industry were announced by members of the Senate. Senator Joseph McCarthy (R.,Wis.) announced that he was considering an investigation into the stockpile program with particular emphasis on minerals. Sen. McCarthy heads the Permanent Committee on Government Operations. The Senator refused to say what mining states he would visit or what minerals were involved. He has already investigated Government stockpiling.

Another investigation will study American access to strategic raw materials with special reference to American self-sufficiency during wartime. It will be conducted by the Sub-committee on Minerals, Materials, and Fuels, headed by Sen. George W. Malone (R.-Nev.). In addition to checking the raw material situation the committee will also check into the "serious situation" in which the American mining industry finds itself. Hearings are to begin late in September.

Recently the House Small Business Committee studied mining industry problems, placing the blame for current troubles on poor planning by the previous administration.

Penn State College Minerals Addition

The new Mineral Science building addition at Pennsylvania State College is scheduled for use at the opening of the fall semester. Building is reported almost complete.

The Baton Construction Corp., Philadelphia, building the addition, erected the original building. Nicklas Shollar Associates of Altoona, Pa., designed the $380,000 addition.

Religious Group Seeks Mine Operation Funds

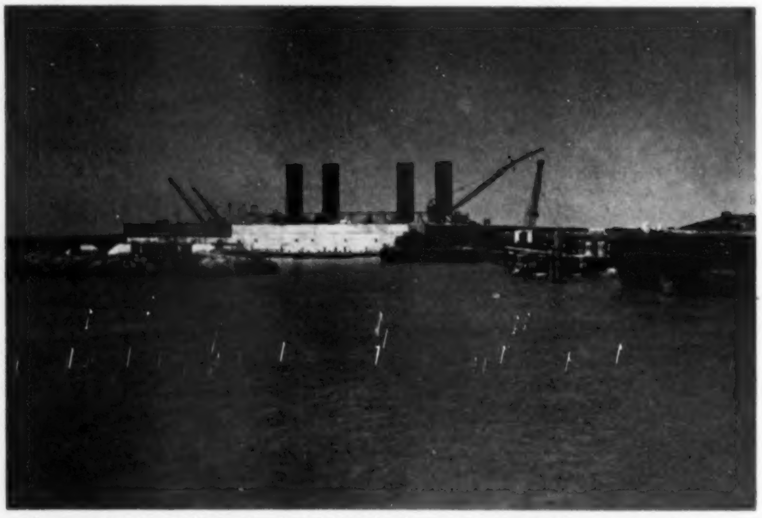
St. Michael's Foundation, non-profit, religious, educational, and charitable organization, has formed the Petaca Mining Corp. to raise funds for operation of the old Petaca mines at La Petaca, N. Mex.

While the mines were known principally for mica, other minerals have been found. An estimated $230,000 will be spent for erection of a mill and other facilities. History of the Petaca mines ranges back to the days of Spanish operation in the 15th century. The mines produced during both World Wars.

Steep Rock "A" Orebody Starts Production

Hogarth mine at Steep Rock Iron Mines, east of Port Arthur, Ont., produced ore for the first time. Ore came from the lake bed. Seventy-five billion gallons of water were pumped from the lake, in addition to 45 million cu yd of clay overburden removed in order to get at the Hogarth mine or "A" orebody as it is called.

The deposit is four miles long and 250 ft average width. Potential reserves at Steep Rock have been estimated at a billion tons.



Freeport Sulphur Co.'s Garden Island Bay development is scheduled to start production early in 1954. Huge stacks are heat reclaimers for preheating mining water. The plant's 16-ft high foundation rests on more than 2000 pilings driven 85 to 95 ft into the marshland. Production is expected to be 500,000 tons of sulphur annually.

Igniter cord is a new device being used at International Nickel Co.'s Sudbury district mines. A miner is putting the igniter cord in place at the Frood-Stobie mine. According to International Nickel, igniter cord requires a blaster to remain in the loaded face only long enough to light a single fuse.

Michigan Tech Offers Additional Scholarships

Two mining engineering scholarships go into effect at Michigan College of Mining and Technology this fall under sponsorship of two members of the industry.

Pickands, Mather & Co. is providing $250 assistance to four freshmen each year, while the Copper Range Co. scholarship provides $2000 assistance to one winner each year for his four years in college.

Pickands, Mather & Co. will select one of the four recipients of its freshman assistance for an additional $1500 for the three remaining years.

Michigan, Minnesota, and Wisconsin residents are eligible for the Pickands, Mather & Co. scholarship, while Michigan residents may receive the Copper Range assistance.

Other industry-sponsored scholarships for study in mineral industries at Michigan Tech are sponsored by Montreal Mining Co., Inland Steel Co., Kennecott Copper Corp., Cleveland-Cliffs Iron Co., and Algoma Ore Properties.

GSA Signs Contract For Champion Output

General Services Administration announced signing of a contract with Copper Range Co. of Boston for delivery of 7,965,000 lb of refined copper by Dec. 31, 1955 at 32¢ per lb.

Ore will come from Copper Range's high-cost Champion mine at Painesdale, Mich. The company is to deliver 1,593,000 lb of refined copper by the end of 1953, and 3,186,000 lb annually in 1954 and 1955.

Expect Marmora Mine To Produce by 1954

Bethlehem Mines Corp., Bethlehem Steel Co. subsidiary, visualizes possible production from its Marmora, Ont., property during the fall of 1954. Earlier predictions foresaw production in 1955.

Stripping operations have been intensified, and first exposure of the orebody was expected shortly. Canadian Turner Construction Co. Ltd., subsidiary of the American firm of the same name, has been awarded a contract for construction of a milling plant at Marmora. Completion of the plant is expected in about a year.

Estimates of the Marmora deposit range up to 19 million tons of ore to a depth of 500 ft. Ore is believed to run even deeper. It has been proposed to mine at a rate of 1 million tons per year.

Uranium Miners Get $2 Million Bonus

Some $2,162,378 was paid to uranium miners in the form of bonus for uranium ores produced from eligible properties since the bonus program for initial production was started March 1, 1951, according to Sheldon P. Wimpfen, manager, Grand Junction operations office, div. of raw materials, Atomic Energy Commission.

Approximately $150,000 a month in payments are being made. The bonus system is aimed at encouraging private industry to locate new deposits and as an aid in deferring initial costs of putting mines into operation.

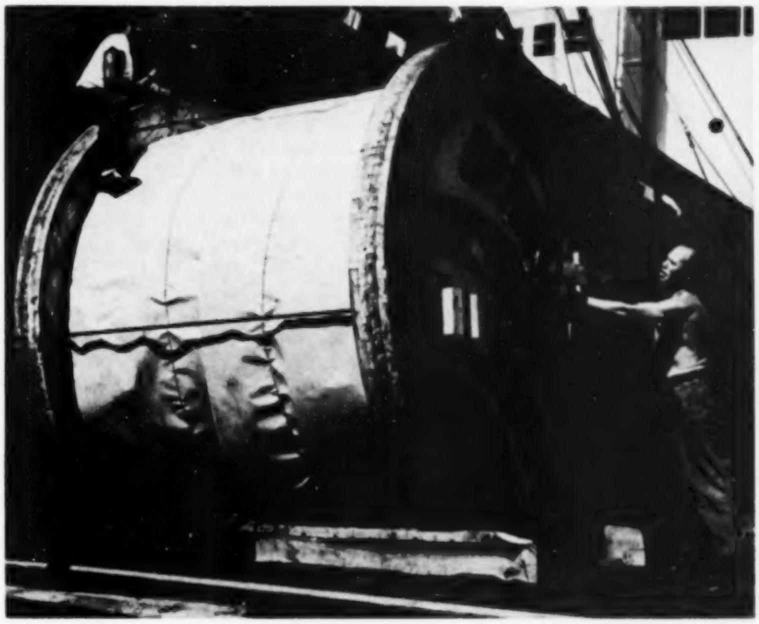
Reynolds Jamaica Mines Ltd. To Have World's Largest Monocable Tramway

Bethlehem Steel Co. shipped one of the longest lengths of 1½ in. wire rope ever wound on one reel to Reynolds Jamaica Mines Ltd. for use in what will be the world's largest monocable tramway.

The reel held 15,500 ft of wire rope made at Bethlehem's Williamsport, Pa., plant. Four other reels, each with 14,000 ft of rope will be shipped to the Jamaica operation. Total shipment will be 71,500 ft.

The monocable tramway will extend from the mine to the shiploading terminal at Ocho Rios. Ore will move downgrade from a valley at 1100 ft altitude. The tramway has two tangent sections. The upper, about 27,000 ft long, requires 56,000 ft of rope including splicing and terminals; the lower, 7,500 ft long, requires 15,500 ft of rope.

To make the longest loop, the four 14,000-ft lengths will be spliced in the field. The 15,500 length will make up the lower section of the tram. The rope rides on idler sheaves placed near the top of 125-ft towers. Bauxite carriers are automatically transferred from one tramway section to another at an angle station. The mine is scheduled to produce some 1 million tons of ore annually for shipment to Gulf ports where Reynolds Metals Co., parent company, maintains reduction plants.



Huge reel containing 15,500 ft of wire cable is loaded on S. S. Cape Cumberland, New York, for shipment to Jamaica. It is part of almost 14 miles of rope for tramway to be installed at Reynolds Jamaica Mines Ltd. operations.

A FEW days before Congress closed shop for the summer, Senators James E. Murray and Mike Mansfield, Montana Democrats, submitted a bill which would set up a minerals agency separate from the Department of the Interior. The bill, S. 2432, has been referred to Sen. Joseph McCarthy's Committee on Government Operations. The Budget Bureau and Interior Department may have their say on the bill sometime soon. The Operations Committee has not decided to hold hearings this fall, but if they do it should prove an excellent sounding board for mining industry opinion.

The bill is comprehensive and probably requires a good deal of consultation before any action could be taken. Senator Murray points out that no specific criticism of the Department of the Interior is meant. However, he feels that the tremendous growth of the minerals industry requires that a separate agency be set up rather than to allow industry affairs to be left in the hands of a branch of the government which is concerned largely with electric power, irrigation, public lands, and Indians. One of the chief motivations for the bill on the part of Senator Murray is the current economic plight of the lead and zinc industry.

This is one of the first concrete actions on a subject which has been broached many times in the last few years. Opinion is divided concerning a separate agency for the minerals industry. Some feel that it rightfully belongs in a department concerned with the matter of public lands. Others are inclined toward belief that only a specialized government office can deal sympathetically and properly with the problems of the industry. Hearings, if they are held, will doubtless result in a cross section of opinion which will help legislators make a decision.

One of the opinions expressed by Senator Murray was that if the mining industry had cabinet representation the current "moribund" state of the mines in the West might have been eased. Interior Secretary Douglas McKay asked Congress earlier this year to reject higher import duties on lead and zinc.

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GEORGI MALENKOV, the Soviet Union's No. 1, tossed a slightly smaller bombshell along with his announcement that Russia now has the H-bomb. He told the Supreme Soviet that USSR steel production will reach 44.7 million tons by 1955. Russian tonnage was a lightweight 1,868,000 metric tons in 1924-25. This year's production will rise to 38 million tons, according to Malenkov.

U. S. production in 1952 was approximately 93 million tons. If reports are to be trusted, production in Poland, Czechoslovakia, Hungary, and Rumania amounted to some 9 million tons. Thus, the combined production of the Iron Curtain countries still remains a long way from that of the U. S. However, if the Russian figures are to be trusted, it indicates the tremendous strides made by the Russians in their economic development.

Mr. Malenkov said that the Russian consumer is to get a larger share of steel production. But along with the promises and the new budget, economists and journalists discerned several hidden items which added up to pretty substantial military expenditures. Economists are questioning Soviet ability to give more to the people and still maintain the military pace it appears to be going.

In contrast with Russian production for 1955, a look at production in the U. S. for the first seven months gives added proof of American leadership. Some 67,229,936 tons of ingots and steel for castings were turned out. It represents the greatest amount ever produced for a similar period in the U. S. and totals 20 million tons more than during the same period in 1952. July was the sixth month of the year in which production exceeded 9 million tons. Production reached more than 10 million tons for the month of March, the first time in history that has occurred.

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AND in reply to the pessimists who decry the dwindling supply of iron ore, Battelle Institute says that future ore deliveries will be more than enough to meet needs. According to Battelle Director Clyde Williams, future domestic dependence will switch from high grade Lake Superior ores to Lake Superior taconites and other low grade ores. Combined with imports, potential 1965-1975 supply will be around 211 million tons, Mr. Williams states. It represents an increase of 54 pct of potential supply expected from all sources during 1950-55. Part of the ore supply for the 1965-75 period will come from Labrador-Quebec mines and Venezuela.

"The complete projection figures also show that iron ore used in the Central Area (Pittsburgh-Youngstown-Lake Erie-Western New York-Detroit-Chicago) would increase only 22 pct between 1950-55 and 1965-75. The Central area would remain the chief steel-producing area. During the same period, however, the consumption of iron ore would be more than doubled in the Eastern, Western, and Gulf steel-producing region, thereby reflecting a trend toward greater decentralization of steel production."

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A REPORT issued in Lusaka, capital of Northern Rhodesia shows clearly that the greatest single source of income for the Central African Federation is its copper. The report indicates that inflation, rather than increased production, brought prosperity to that part of the world. Value of mineral production during 1943 to 1952 rose from £13 million to £79 million, while the increased copper tonnage was comparatively small.

This would indicate that American price trends can have a tremendous effect upon Northern Rhodesian economy. Should the American copper price go down even further, as some expect, the Federation

would suffer. It exists under an inflation which would take a bad tumble should U. S. markets drop. The decline would have the effect of shoring up gold prices, however, and to some extent Southern Rhodesian gold production would ease the situation.

MURIEL Sibell Wolle has written a book that ranks with the best of that growing literature which attempts to preserve the glory, trials, battles, and wonders of those days when men went into the west seeking gold. They were hard men, living hard lives in one of the toughest eras in American history. Many of them forgot morals, law, and justice in their search for riches. They pushed back the Indians, who in turn retaliated in a battle for life.

Their monument lies along the "Bonanza Trail", that aimless line that can be drawn from strike to strike. The memorial consists of towns without people, buildings that house only memories, and empty holes in the ground. The story told by Mrs. Wolle's book, *The Bonanza Trail*, is rich in history of a kind not taught in classrooms. Removing its outer trappings discloses the striving of a people for something they perhaps only partially understood.

The outstanding thing about the book is the many sketches of towns which were once the sites of mining booms. Silver City, Goldfield, Sparta, Homestead, Frenchtown Bar, Kettle Falls, Leesburgh, Alta, bring forth a picture painted in bold strokes on a broad canvas. By sketches and words they seem to come alive for a moment.

Goldfield, where Tex Rickard established the Northern Saloon with 80 bartenders, was one of the more fantastic towns of that era. It was Rickard who staged the fight between Gans and Nelson for the lightweight championship of the world. Gans won after 42 gruelling rounds. After the crest of 1910 and 1911, Goldfield declined. Today, the railroad station is empty, the saloon silent. There are a few people who live there, but the Goldfield Hotel is closed. Only the Florence mine has reopened.

Mrs. Wolle traveled some 70,000 miles by car and on foot to get her material. She talked to hundreds, possibly thousands of people, driving over good roads, bad ones, and sometimes over trails that hung over the side of cliffs.

She discovered stories like the one about Russian Bill, a self-styled badman who ended up at the end of a rope after shooting off a finger of some innocent person during a moment of high spirits. Then there is the tale of the miner who chased his wandering mule into the hills and reached for a rock to throw at the peripatetic beast. Glancing down at the rock the prospector decided to temper his anger somewhat—the rock contained gold. Some of the mining towns have survived. Today, they are respectable law-abiding places, with industry often crowding out the past. Here and there buildings still exist that hail back to a more rambunctious day. Mrs. Wolle succeeds in capturing a magnificent, bawdy, and adventurous era in American history that is a far cry from the highly mechanized mining ventures of today.

THE coal industry has come up with an idea that may bring coal and electric utilities substantial new business and at the same time, save the steel industry millions of dollars annually. After two years of research, Bituminous Coal Research Inc., in conjunction with 14 electric utility firms says that the electric furnace should replace the open hearth for steel production. Bituminous Coal Research says that the results of investigations show that savings of up to $3.15 per ton in making low carbon steel from cold metal are possible. Key to the proposal is that conversion would raise national electricity production some 12 pct while increasing coal production about 25 million tons per year.

Electric and coal production increases assume total replacement of the 950 open hearth furnaces by 760 electric furnaces. However, Bituminous Coal Research points out that even 10 pct replacement in the near future would be of tremendous importance to the industries concerned. About 83 million tons or 89 pct of the steel made in the U. S. comes from open hearths. Most of this is low carbon.

Three plant sizes were included in the study—250,000 tons, 500,000 tons and 1,000,000 tons annual steel production. No plant more than eight years old was included in the investigation. Three steel industry representatives advised on the collection of data and preparation of the final report. Battelle Memorial Institute conducted the research for the electric companies and Bituminous Coal Research.

The report shows that capital cost for electric furnace installation is only 60 pct of that for open hearths; the cold melt steelmaking process using scrap and pig iron shows lower cost for the electric furnace, and electric furnaces show a greater annual return on invested capital. For the 50 pct hot metal/50 pct scrap process, annual return for the electric furnace is equal to or greater than the open hearth, according to the report.

Electric furnaces today are used mainly for the production of special alloy steels. The World War II expansion left a surplus of electric furnaces after the end of hostilities, leading some manufacturers to experiment with the production of low carbon steels. Improvements, such as the swing roof, high rates of energy input, and increase in furnace size took place at about the same time. The resulting important economies brought electric furnace steelmaking costs down to the level of the open hearth.

The report states that the electric furnace, in addition to capital investment savings and reduced cold melt practice production costs, offers flexibility. It can be placed in production or withdrawn almost at will. Electric furnaces can operate all but 15 days per year, while open hearths are usually down about 30 days. Rebuilding time for the electric furnace is shorter. The same amount of materials yields about 2 pct more in the electric furnace. Sulphur control is greater in the electric furnace. More accurate temperature control saves steel production time. Nitrogen control by the electric furnace is expected to match that of the open hearth. The report can be obtained at $10.00 per copy from Bituminous Coal Research, Inc., 2609 First National Bank Bldg., Pittsburgh, 22.

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Drift of Things

BY the time you read this column, the members of the Mining, Geology, and Geophysics Div. will have received a copy of the first newsletter distributed by the youngest and largest division of the Institute. The newsletter is not a new technique among the divisions of the Mining Branch, for at least two others have used it successfully to bring important news to the attention of their members. First of all this booklet tells you that you are a member of this division, which in itself is an important part of the story.

Then it gives you news which is vital to you as a member of the Institute and an affiliate of the Division. Programs of the divisional fall meeting and general activities are provided as a supplement to tne information appearing monthly in *ME*.

The officers of the Mining, Geology, and Geophysics Div. responsible for this publication feel that the newsletter advances the work of the group and stimulates allegiance to the division. Your division provides the mechanism for participating in the society activities on the national scale, just as the Local Section cares for the local needs. This division publication will be the newspaper of your national section.

ONLY a short time back we were approached by a Local Section officer while traveling in his area, and he questioned us on the eligibility requirements for members of the Institute. The particular case was a mechanical engineer employed by a mining company for design work and shop direction in the mechanical department. Since that time, it has become evident that this problem comes up quite often before the various membership committees of the Local Sections, as well as of the Divisions, so we will pass along the explanation given the Section Officer.

Any person actively engaged in the mineral industry on a professional level is eligible for membership in one of the three classifications, i.e. member, junior member, or associate member. College education is not a prerequisite, nor does the fact that the candidate is a mechanical, electrical, civil, or other engineer, jeopardize his eligibility. The junior membership and full membership are only grades as to experience and professional status. Associate membership is for those persons closely connected with the industry, but not actively engaged in technical direction.

SOME do and some don't. When mining people get together, stories of prospectors, their foibles and their luck, increase unwritten mining lore. The press rarely gets the stories, but recently two of the West's hopefuls broke into print.

Time and *Business Week* both told of one fellow who hit bottom, down to the beans three times-a-day level, but he and his wife and children stuck together and kept trying. Some of "Hard Luck" Charlie Steen's friends believed in him when his drill rig broke down just as it had hit high grade uranium ore. This year Utex Exploration Co. shipments have hit the million dollar mark. Charlie drives a flaming red Lincoln now.

Newspapers in New York caught the story of the other prospector—who had to get on the TV show of the same name to "Strike It Rich." Hubert Miller has been trying for most of his 94 years, but that hasn't dulled his hopes. After he hitch-hiked to New York from Auburn, Calif., a local hospital donated a cataract operation. His hopes haven't dimmed, he'd like to get a grubstake and have another try.

Moral of the stories, if any, probably is that the true prospector is the one who never quits.—R.A.B.

A RECENT editorial in *The Saturday Review* revealed that readers are not the only ones disturbed and puzzled by the peculiar picture of America painted by our authors and proclaimed as the "true image."

"Too many of our younger writers have inherited from the years of the Great Depression in which they grew up their lack of belief in their own country and in the dignity of man; and they are now selling short not only human nature but America itself."

"For the 'social consciousness' that produced in the late Thirties so many bad books . . . they have substituted the sterile theme of human depravity and hopelessness borrowed from Sartre and Kafka."

Thus, Europe, which at one time looked to the U.S. as the "one bright, if hectic, spot in the midst of dismal and apparently endless world depression," today sees us through our writers as a nation with power, but which wastes that power in doubts and neuroses.

But the editorial does offer hope that it has become time for a change. "It represents a curious and perverse interlude in our long literary history, and it is already vanishing since the public and the publishers are the final arbiters and they have had quite enough of perversity and unreality."

IN looking back over the summer, we would be amiss not to mention the Coronation of Queen Elizabeth II, and the conquering of Mt. Everest. One minor result of the Coronation was the appearance of a photograph of the Queen on the cover of the June issue of the *Canadian Mining & Metallurgical Bulletin*. To our knowledge this marked the first appearance of a "cover girl" on a technical magazine, and it was a welcome sight indeed. However, we have since been informed by Ed Robie that the August 1936 and July 1946 issues of MINING & METALLURGY featured "cover girls." Just shows that can't beat AIME!—A. S. C.

PRINTERS INK recently carried the following chuckle: "A friend of ours burned a hole in his new summer suit made of a combination of the new synthetic fibers. Didn't know whether to take it to a tailor or a chemist."

Charles M. Cooley

From Ore Testing to Cost Estimation

A Mill Design Symposium

Chairman S. D. Michaelson, Tennessee Coal & Iron Div., U. S. Steel Corp.

Associate Chairman R. M. P. Hamilton, General Engineering Co., Toronto, Ont.

PRACTICAL PROBLEMS AND BASIC PRINCIPLES *ranging from ore testing to cost estimation get a working over from top men in the minerals beneficiation field. The Minerals Beneficiation Division developed this symposium for presentation at the 1953 Los Angeles General Meeting to assist the many people who think that having a new plant necessitates a lot of trouble and dismay. The papers and free discussion from the floor give ways and means of approaching a new plant that will relieve the strain on the organization, if adequate attention is put on the program before work begins — that is if thought is given to the program itself, before tackling the actual mill design.*

Ore Testing to Determine Flowsheet Prior To Plant Design

by Bruce Irwin, V. L. Mattson, Gene Meyer & S. Power Warren

WHY an ore testing program? Too often the test program ends when determination of economic grind required to unlock valuable mineral has been supplemented by a few laboratory scale metallurgical tests. As a careful and methodical study of the physical features of the ore produces detailed information on mineral and metal relationships, the need for a complete program becomes evident. Factors appearing insignificant may be of great economic importance if they affect filtration rate, water recovery, or tailing disposal.

Testing should start **WHEN** development has produced adequate quantities of representative ore and reserves justify the project. Basic considerations of mill design require at least a basic flowsheet, but there is no surer way of wasting money and time than to conduct tests on ore that is not representative of mine production.

WHERE should testwork be done? Ideal facilities would be equipped with microscopic, spectroscopic, and other analytical equipment and various special laboratories. The pilot plant would be adjacent, the mine would be close by, and water that would be used in the final plant would be piped in.

If the company has operations in the same district, these facilities could be used. Even then, it may be advisable to consider an unbiased approach by an outside research organization.

Existing mill laboratories can seldom provide highly specialized equipment. To provide research facilities needed for modern ore testing, a number of mineral industry research organizations have been developed with staffs of both practical and scientific men.

HOW should the program develop the most effective and economical flowsheet? A clear and accurate statement of the objective of the program is necessary. It is easy to ramble into inviting but unessential paths of investigation.

Contractural relations between the sponsor of an ore testing program and a laboratory organization

V. L. MATTSON is Director; BRUCE IRWIN, Research Supervisor; GENE MEYER, Ore Dressing Supervisor, Colorado School of Mines Research Foundation Inc. S. POWER WARREN is a consultant, Lakewood, Colo.

The Program:

1 The first step in the program is a survey of relevant published information, and a cursory patent search.

2 The all-important step, analysis of ore, may normally be conducted at the same time as the literature search. By analysis, much more is implied than determination of chemical composition. What minerals are present and a picture of their relations to each other is required. Providing this information is primarily the job of the petrographer. Spectroscopic examination and X-ray diffraction studies are useful. This investigation may also set the analytical procedure to be used throughout testing.

3 The analytical phase has established minerals to be recovered, size of grind to be used. Magnetic susceptibility and other physical properties of all minerals are in the hands of the mineral dressing engineer. The method of concentration has generally been decided up by this stage and a few simple tests at this point narrow the field of the actual testwork.

4 Size of grind has been determined. Satisfactory reagent combination has been established. Effect of slimes on reagent consumption has been studied. Preliminary information has been obtained on settling and filtration rates. Importance of water consumption and quality is known; pH at all circuit points has been measured.

5 In the pilot plant operating problems of the final plant begin to materialize. True rates of reagent consumption are revealed after hours of operation. If a recirculated middling builds up, requirements for regrinding or a separate circuit become apparent. Filtering rates may look different at the end of a 24-hr run. It is safe to say that no mill is small enough to justify by-passing of pilot scale testing for establishing final flowsheet.

should be detailed and the objective, time limit, and financial limitations should be clearly stated. Legal protection of new ideas, certification of laboratory notes, procedure for patent application, and ownership of rights should all be considered in the contract, as well as the status of operating company personnel who desire to observe test work.

The report of the ore testing program should pinpoint the final mill flowsheet. But it should also do much more. A well-designed and executed program would present all data for mill design and furnish data on economic and financial phases.

Unless there is a single agent responsible for coordinating activities, trouble is almost certain. When a contract is let for design and construction of a chemical plant or a petroleum refinery it is usually on a *turn-key* basis with an operating guarantee. A single responsibility exists from testwork to initial operation. Ore dressing plant problems are not exactly parallel, but more thought should be given to the need for single responsibility.

Just as the flowsheet is the basis for mill design, so the ore testing program serves as the basis for the design of the flowsheet.

Chairman Michaelson: I think that Mr. Irwin has provided us with a springboard from which we can dive into our discussion.

E. H. Rose: (*Tennessee Coal & Iron Div., U.S. Steel Co.*) The outline of ore testing was based on a premise that our conventional processes and equipment will be put into some configuration which will apply to this new ore. Be it San Manuel, be it White Pine, shall we put together our old tools in some new configuration, or will we have more courage than that?

What would you do with a taconite if you had never treated it before? We tested all the different methods that we have, but they weren't quite good enough. Basic research laboratories sprang up all over the Iron Range. Concentrate wouldn't go into a blast furnace and stay there. The laboratories had to come up with a new process for conglomeration —pelletizing. The magnetic taconites are amenable to conventional methods of treatment, but here you have not heard anyone tell you about processing nonmagnetic taconites.

On the nonmagnetic taconites we possibly could go to roasting, hematite made magnetic by artificial means. The process was described in the AIME *Journal* in 1892. No plant in the U. S. today is concentrating hematite by means of a magnetic roast. Why? Low grade iron ore is a pretty cheap commodity. Most roasting processes are fundamentally inefficient and it was not until new methods were established that magnetic roasting of iron ore could be developed. Where were these discoveries made? Heat exchange in the chemical industry and Fluo-Solids roasting in the oil industry. We can go into other industries and find discoveries and lessons we can apply.

Jack Myers: (*Consulting Engineer.*) I think a springboard into the unknown is purely wishful thinking. After all, the people who are responsible for spending $8 or $9 million on a new milling program usually have reached an age where obtaining a new job is very difficult. The matter of self preservation simply dictates that they play it very close to their vest. A plant went into operation this year involving many millions of dollars. That plant was 15 years old before it ever turned a wheel, because they didn't do anything that was even the shadow of a chance. . . . There are other cases. This isn't to be criticized. The history of our development of civilization is a slow, tedious, process.

Dr. Le Baron: (*International Minerals & Chemical Corp.*) In the nonmetallics industry the price of the final product has not gone up with the inflation of the dollar. It has gone up, but not to the same degree as everything else. . . . There are plants operating today, not too efficiently, but at a nice profit. Construction costs have gone up so much that if operations making profits today were rebuilt on no grander scale, they would operate at a loss. That is lending impetus to development. I think one of the most fundamental things about new mill design is not the laboratory, but the boys who are on the boards really designing a plant which is economical, fundamental, at a low cost.

Robert Crockett: (*Consulting Engineer*) In my work I am always faced with the necessity of meeting the economic situation. Progress is a matter of growth. I base my work on those processes which exist and have proven practical, always looking to the laboratory man to add something to that which already exists.

Norman Weiss: (*American Smelting & Refining Co.*) In defense of myself and others, and referring to what Jack Myers said, we can't afford to gamble. Our jobs at the moment will total close to $40 million. Experimentation belongs in laboratory and pilot plant. . . . Successful plans of the past few years departed only slightly from established lines, but in some departing they established a new pattern. Now, to step further away than that and spend a lot of money would be suicide, not only for me but for the whole profession.

To refer back to one point in Mr. Irwin's paper, in the chemical industry, a few of you have kept in mind that the pilot plant is not built to prove the process, but to prove instrumentation. We build a pilot plant to test the process and very rarely think of putting in full scale instrumentation.

E. H. Rose: It is perfectly true that we cannot step out and depend entirely on innovation—nor did I say that. I only said that while we make every penny go as far as we can, using the tried and true methods, we must retain courage, imagination. . . . keeping in mind that we are not going to be doing everything next year exactly as we did last year.

B. A. Reak: (*Roberts & Schaffer Co.*) Here is a point of view on the situation when a flowsheet is handed to the engineering contracting firm. They say, "Here's what we want." I think you have to know a lot more about the plant than the flowsheet, to find out what other processes are being considered. . . . You work on it a month or two, and they will say, "Here's one point back here we want to change." You do a layout to get the smallest plant you can, establishing various points, and they want to back to the start and change something. We are glad to change it, but it costs money. An engineering firm would like to have the flowsheet set up without changes because you have given a figure on plant cost and by the time all the changes are made that figure is doubled.

Fundamentals of Mill Design

by H. L. McNeill

THE ore body has been sampled; the test work done; and all the data we need are on hand. Remember, one false assumption here will be reflected in the cost of the finished product for years to come.

So, it is fundamental that one must know the ore, i.e., see it in place, as it is broken, handle it, feel it, know whether changing seasons affect it. The same physical study must be carried out on all the products that were made during the test work and a careful study made of other raw materials which will be required for processing. The ore must be handled at the lowest cost possible

With this behind, site selection is next. Power, water, tailing disposal, favorable grades, prevailing winds, natural drainage and character of the soil, or rock, must be studied carefully. Here again, the cost of the finished product may be influenced adversely by a poor decision. Particular attention must be given to possible plant expansion.

Selection of Machinery

Now the machinery can be considered, eliminating many variations of car dumpers, feeders, crushers and conveyors. The capacities of these machines and whether it is safe to use a single machine with inherent economies in building space and operating cost is the consideration. If the ore is hard and abrasive, multiple machines may be put in open circuit; if it is soft, fewer stages and possibly some closed circuits will be used.

The individual function of each machine must be studied in detail, proper clearances must be provided so maintenance and repairs may be made with facility and with the fewest number of man-hours possible.

While overhead cranes may be idle over 90 pct of the time, they can easily pay for themselves during construction and later be used to keep operating costs down.

Lowering Labor Costs

It is time to think of operations which can naturally be grouped together and see if these machines can all be operated from one level. Provide platforms which give a commanding view over the plant and few operators will be required.

A study of the job classifications of the district will sometimes show that new machinery operated by one man can be used in place of the conventional machines which may require men of more than one classification. While the new classification rate may be higher, it is almost certain that it will be lower than the combined rates of the other. Consider the use of man-lifts at strategic points if their use will reduce the operating force.

With fewer men, the plant and the yard must be well-lighted; 1.5 to 2.5 watts per sq ft for inside lights. It must be safe and easy to keep clean. Instrumentation and the proper controls become increasingly important.

H. L. McNEILL is an engineer with the Mining Dept. of Stearns-Roger Mfg. Co., Denver, Colo. This paper was presented at the symposium by MR. BEACH.

The work required to get going again after a plug-up can be reduced if certain machines in a sequence are properly interlocked, or furnished with shut-down alarms. In certain places, the Utiliscope, television's gift to the operator, will eliminate a man and insure continuity of operation.

In any event, when the arrangement is complete, check with the machinery manufacturer for a final go-around. Remember in the final analysis, nothing can be more fatal than one false assumption.

Design Considerations

Now that all of the machinery has been determined and arranged, launders, pulp piping, water piping, duct work for dust collection, the electrical conduits and compressed air lines and fire lines must enter the picture. These items are mentioned in order of priority. Launders in general have right-of-way over everything else, followed by pulp lines; electric conduits can be run almost anywhere and have a low priority.

At this point, a scale model may be invaluable. It is almost impossible for one group of draftsmen in charge of piping, another group on electrical work, another on duct work, not to run afoul of each other without a scale model.

Launders must be sloped conservatively, and have the right cross section to allow for surges when starting up, or shutting down; ample junction boxes should be provided at the turns, and dead boxes to avoid wear where there is a drop. Many types of lining are available, in this study the effect of reagents should not be overlooked. Pulp lines must be accurately sized to provide the correct velocity and any bends should have large radius and be installed so that worn bends may be easily replaced. Long tailings lines should be kept on grade even though this may require expensive trestles at some points; otherwise hillside contours may be followed.

Mill water mains should be so sized that one section may be started up, or shut down without unbalancing the section on either side.

When pipe lines have to be buried, soil tests should be made to determine whether cathodic grounding of the pipelines is necessary. Cathodic grounding provides a one volt potential in the pipe above the surrounding ground, so that current flow will be into the pipe causing deposits to build up on the pipe. If flow is reversed, the iron molecules of the pipe are deposited in the ground and leaks may develop.

No plant today can afford to overlook the adequate handling of dust. First of all, there is the health hazard for the men working in the plant, and the number of law suits which have been filed because of dust nuisance in a community. Secondly, manpower comes into the picture again for clean-up. Some dusts are magnetic or become magnetized in the field surrounding electrical equipment.

In the north country, dust collectors must be efficient enough so that the clean air may be returned to the room, otherwise the heating problem would be beyond all reason. It is possible to use the clean air to cool the force-ventilated motors.

Electrical Transmission

If possible, the electric transmission line should skirt the plant site with the shortest distance to the main substation. It should be carried on high poles where it comes through the site, so that moving equipment, such as cranes, will not interfere with the wires.

The main substation should be located adjacent to the main switch gear room. Power coming to the main substation will normally be 69 or 110 kv and this will be transformed to either 2400 or 4160 v for general plant power distribution.

At the load centers, which are located to provide short runs to areas where motors are concentrated, the power is transformed through non-inflammable oil, or air cooled type transformers to 480 v for distribution to the small motors and general plant power requirements. Further distribution is made through the motor control centers which have a group of combination type motor starters for the centralized control of a group of related motors. Circuit breakers which feed lighting transformers, welding receptacles, vulcanizing outlets and electric heating units, are also provided for these motor control centers. The lighting transformers feed multi-circuit lighting panel boards.

Push button control panels are located where the operators will have a commanding view of the equipment they control. These panels should also provide load indicating ammeters, alarm circuits, and equipment running lights. Since these panels are out of sight of some of the equipment which has to be started, they are provided with a loud alarm siren, which has to be sounded before any piece of equipment is started. Stop buttons at the panels and adjacent the motors, can be locked out during repairs, also a positive circuit shut down should be provided at the motor control centers.

Because modern plants are planned to operate with the fewest number of men possible, the use of stand-by emergency power and emergency lights is recommended. If they are properly installed, they will come on the line automatically during a power failure, and start picking up the load in ten seconds. This emergency power can be used to keep certain machines operating which otherwise would require a long period to wash out before they could be restarted. It provides for emergency lights at the heads of all stairways, landings and adjacent the machines which will keep operating.

The primary transformers are provided with resistor grounds which will limit the voltage on the high voltage motor frames to approximately 100 v. All machines and all buildings are grounded through a grid to a well, or some positive ground, in the interest of safety and electrical continuity.

Buildings

A complete process with the machinery is arranged and it should stand to reason that it will be adequately housed for climate. A few points are fundamental:

1 Check steel designers on sway bracing and wherever possible, brace to the various floors and supporting structures. This will leave more room for stairways and platforms.

2 In planning the different floor levels, leave ample headroom since most of the conduit runs, pipelines, and duct work are normally suspended overhead. These have to be fitted in long after the steel has been designed and purchased.

3 Use floor grating wherever possible. In this way only the basement floor will need cleaning. In the basement leave ample slope for washing down.

4 For the last ten years, windows have been going out of style, their upkeep and heat loss is high. No one ever turns the plant lights off during the day time, and it is now recognized that glass windows, as a source of operating light, are a waste of money.

5 Ventilation can be efficiently accomplished with power driven roof ventilators, or sliding doors at the different levels.

6 In planning the buildings, it is frequently desirable to locate shops and warehouse in the dead end of the operating building; leaving the other end of the building for plant expansion.

7 Railroad cars can then come into one end of the building and be unloaded with the same overhead cranes that serve the operating machines and shop area.

8 Do not overlook the warehouse; some plants require very large areas. These should be fenced off with chain link wire.

9 Since a large operation is to be conducted smoothly by a few men, there must be an efficient system of plant communications. Inside dial phone systems have been developed that are entirely automatic.

10 One other fundamental that has much to do with the cost of designing and constructing a plant is the thrashing out of certain controversial issues. After management has given them due consideration, the decisions should remain unchanged.

Mr. Weiss: I have very few comments to make on this paper. It is extremely practical, that's the main thing about it, and you can tell by it that Harry McNeill has had a lot of experience in building plants and knows where sources of trouble are. The last line, item 10 above, is the one that carries the punch. It is certainly very important and if you don't follow that procedure you are bound to get into trouble. Each organization should, if possible, reduce the number of channels to one at each end. It is a good way to reduce expenses and keep out of trouble.

Mill Design

The Value of Ore-Bedding Systems

by Warren L. Howes

UNLIKE crushing, grinding, and concentrating equipment, the value of ore-bedding systems is not readily apparent, due to scarcity of comparative results between various ore-bedding systems within an existing plant. Too often ore-bedding systems are thrown into project plans as an afterthought and frequently are the first to suffer curtailment in the ultimate drive for economy.

Ore-bedding plays an important role in plant production, efficiency, and operating cost. Yet, expenditure for ore-bedding systems rarely amounts to more than 10 pct of concentrating plant capital cost. With as much as 25 pct of the inherent plant capacity at stake, selection of the ore-bedding system should be considered on an equal basis with the crushing, grinding, and concentrating equipment.

Every plant has ore-bedding problems peculiar to itself, but from general considerations the system should serve the following essential functions: 1—absorb surges in tonnage, character, and grade of ore between mine, crushing plant, and mill, 2—smooth out flow and segregation between crushing stages, and 3—provide steady and uniform feed, both in character and grade, to the concentrator.

Surge bin capacity between crushing stages is often neglected in plant designs. Crushers work best under constant load at capacity. Few existing plants with two, three, or four crushing stages have all machines running at capacity. Usually one type of crusher becomes a bottleneck and the rest of the stages run underloaded to accommodate it. The solution is to provide adequate storage ahead of each stage so that changes in ore and tonnage can be absorbed and each crusher can produce a maximum.

Crushing may consume 20 pct of total power used in crushing and grinding. This sizeable block of power should be utilized to the fullest extent.

Fine ore bins between crushing plant and concentrator are common practice, but methods of bedding vary greatly. The system should provide: 1—adequate supply ore for the concentrator while mine or crushing plant are shut down, 2—uniform character of ore to the grinding circuit, and 3—uniform grade of ore to concentrating section.

The basic problem in ore-bedding lies in combating segregation. Fines inherently concentrate below the loading point of the bin, while progressively coarser material rolls down the slope of the cone toward the perimeter. This automatically segregates the ore, with fines in the center of the bin and the coarsest fraction around the edge.

When material so bedded is drawn, however, it glory-holes vertically to the surface until the cone of material has inverted with sloughing inwardly. The result is violent fluctuation in character of material drawn by any feeder.

Actually two entirely different types of concentrators would be required to treat the fines and the coarse material separately at the same rate from a given ore. The problem is to make one concentrator do the job efficiently.

WARREN L. HOWES is Hibbing, Minn., manager of Western Machinery Co.

Whether the system be for run-of-mine, intermediate crushing stages, or fine ore storage, ore-bedding systems can be classified as: 1—single-loading, single-draw, 2—single-loading, multiple-draw, or 3—multiple-loading, multiple-draw methods. Generally speaking, the more loading points and the more draw points in an ore-bedding system, the better segregation can be combated, although expense increases.

In crushing plant surge bins, variable speed feeders, singly loaded bins with single draws, and several hours live storage are sufficient. In fine ore bins, however, it is the writer's opinion that no ore-bedding system is adequate with less than multiple loading and multiple draw, the loading being in sequence and the drawing being done continuously.

The point of greatest profit in ore-bedding systems is in the fine ore storage ahead of concentrators, with dividends in: 1—more production, 2—higher concentrate grade, 3—reduced tailing losses, and 4—lower operating costs.

Comparative results with various types of ore-bedding systems are a rarity. However, where without any other change, fine ore storage was switched from a single-loading, single-draw system to multiple-loading, multiple-draw, the results were: 1—a 15 pct increase in concentrator capacity, 2—an almost proportional decrease in mill costs, and 3—approximately 3 pct increase in recovery.

This concentrator used gravity and flotation methods, operating three shifts, seven days a week. The mine and crushing plant operated two shifts, six days a week.

Added capacity was the result of uniform feed to the grinding sections, enabling operators to load mills to maximum capacity without choking. Ball charges were adjusted exactly, with no surplus of large balls needed to handle surges of coarse feed. Added tonnage was handled with no increase in manpower and very slight increase in power consumption. There were no fluctuations in tonnage, grade, and character of feed to flotation, and operators could utilize reagents to best advantage and control concentrate grade and recovery.

All of these factors added up to approximately 19 pct increase in production, with little increase in cost other than the capital cost of the fine ore bin with multiple loading and draw points. It is estimated that the bin paid for itself in a few months.

Chairman Michaelson: I asked our operating people to look at the bins in two of our plants. One coal plant has a very large bin, a size blender so to speak. The other problem is uniform feed to furnaces so we have a plant for chemical blending.

W. S. Springer: (Speaking of *TCI's* Concord coal plant.) We are blending for uniform size because we split raw feed at ¼ in. by wet screening and treat it in separate processes. . . . The 6000-ton blending bin is in three parallel rows, eight bins to the row. Each bin is subdivided into four parts in the upper section, but the bottom hopper is common to all four. Under the bin eight feeder belts come in at right angles to the three parallel lines. Three gates

feed each belt, one being operated at a time. . . In drawing down it doesn't glory-hole through in each pocket, but breaks through toward the center partition. . . A series of samples showed that the large sizes, rather than increasing out to the periphery, actually decreased — finer sizes showed practically no change at all. Evidently it is in the bin design. Each pocket of the bin is 9 ft square, and the shuttle belt feeding each bin is 6 ft wide, so actually you have quite a volume going in there and little space for it to roll to the periphery . . . Samples were taken by automatic sampler at the top of the collector belt after all feeder belts had fed on, until a particular row had been emptied and showed relatively uniform size distribution.

C. E. Lacy: The plant where we condition all iron ore used in TCI blast furnaces in the Birmingham area was built in 1941 and has had 62 million tons through it. Blast furnace ore is crushed, screened into coarse, medium, and fine sizes, and blended to uniform chemistry.

At present ore comes from six TCI mines, plus foreign and domestic shippers. Output from a given mine has fairly uniform analysis from day to day, but wide variations occur from mine to mine. Southeast area ore has low iron, 34 pct Fe and excess lime, making it better than self-fluxing. Ore in the central area contains up to 40 pct Fe, but 6 to 8 pct excess acidity. Northeast ore drops back to 36 pct Fe, and 8 to 10 pct excess acidity. Up to eight high grade ores are blended with low grade ore to meet furnace requirements.

Facilities are identical for both coarse and medium ore. . . Blending of sinter feed is somewhat different from the blending of ores. . . After crushing, screening, and sampling, conveyors discharge coarse ore into 24 silos, each 67 ft high with 820 tons live storage capacity. Silos are arranged in two parallel lines, divided in six groups of four silos each. Six constant weight feeders for the 24 coarse ore silos permit up to six ores in a given blend. Feeders discharge to a common collecting conveyor and ore is mixed at transfer points and as it discharges to hoppers and cars.

Outside storage facilities are also provided for coarse and medium ore, but are used only to take care of exceptional surges in consumption. An overhead belt and an oscillating tripper spread ore in a thin stream over the length of the pile; reclaiming is done through gates in a tunnel, below.

To show what we are trying to do, for one month, February 1952, we were attempting to blend to 39 pct, and we got 39.02 pct metallic. We were to get acidity of 4.95 and got 4.91. . . Now, what this means to us and the people at the furnaces: If a furnace goes down, a few more cars of high grade ore through the plant produce enough hot metal to take care of that furnace being out. If we can't get lime we need. . . we send low acidity ores to the furnaces and keep them in operation for considerable periods.

Chairman Michaelson: Maybe some of the copper men would care to talk about blending. . . to keep mills and float circuits in balance? (Comment from the floor.) Recently in asking around we found many operators aren't fully aware of what lack of proper blending is doing. So I would like to echo Mr. Howes' remark that it is a phase that should be looked into critically.

W. L. McMorris: (Speaking of *U. S. Steel Co.* coal handling facilities.) We blend coal in bins very

similar to those Mr. Springer described (TCI) and confirm that there is little segregation appearing at the top of the pile. We attribute it to the fact that the cross sectional area is so small in relation to tonnage input. We hoist coal to blending pockets at about 2800 tph. One cell holds a little over 100 tons—there isn't a lot of time for segregation. With cells very deep in relation to cross section there is only 2 pct segregation in coal withdrawn from a cell until we are within the bottom eighth. To avoid this segregation we do not draw one row completely empty, but rather have half of that row half full when the other half empties.

At least 90 pct and more often in excess of 95 pct of determinations are within plus or minus 5 pct of the average. If the percentage $-\frac{1}{4}$ in. is normally 30 pct, 90 pct or more of our samples lie within $1\frac{1}{2}$ pct of that figure. This is important because we too have separate cleaning processes for coarse and fine coals. From the chemistry standpoint it is important. . . we strive for uniformity of sulphur content, despite a terrific spread from 1.3 to 2.7 pct sulphur. On sulphur we maintain that same degree of uniformity, 90 pct of samples within 5 pct of average value after cleaning. I don't think we could come close to producing results asked of us if we hadn't put in an elaborate blending system.

W. S. Springer: In the (TCI) operation we do not empty bins either, we draw them down to where the partition ends and leave the hoppers full.

Mr. Peterson: I was wondering about maintenance for the past 65 million tons in bins of that type from the standpoint of cutting down cell size to approach the chute.

Chairman Michaelson: Talking about 18-ft diam bins in 14 years now, we started with $\frac{3}{8}$ in. plate, no liners, and that is about half worn. Rivet heads flushed out pretty fast. They have been replaced and we are now going through those bins and rebuilding where they need it. **C. E. Lacy:** These are the silos now, and I think we are getting more wear from ore as it falls in there. . . not from sliding down in the bins. When those silos are run empty you have a drop of 67 ft, and all of it doesn't hit the middle.

Question: I was thinking about bins with 9-ft sides. **Chairman Michaelson:** There (with coal), you don't have the abrasion problem you have at Anaconda. We are mining something that runs close to 35 pct rock at that point in the flowsheet, and while that rock is relatively abrasive and up to 2-in. size, it isn't nearly as bad as the other materials. As far as I know we have had no wear in 5 million tons so far. . . Some of the stiffener angles should show wear first, but at last inspection looked all right. They ought to be good for 30 or 40 million tons at least.

Mill Design

Methods of Estimating Plant Construction Cost

by H. V. Hughes

ESTIMATING vitally affects everyone in industry for important decisions are made on this basis. Answers as to whether to proceed with a project, who will do the work, financing, and others are the result. Soundness of these decisions depends on the accuracy of the estimate. Profits are generally the reward of good estimating, and losses may result from the reverse.

Once management decides to investigate a plant installation or expansion, someone poses the question, "How much will it cost?" At this point the project becomes an engineering problem and plant engineers or an independent concern prepare estimates. It must be decided to what extent and depth the estimate will be prepared, what will be the end use, and to establish the degree of accuracy. Assuming that a shotgun or a flash estimate is all that is required at the moment, the information used can be meager, and the time involved can be hours, or at most, three or four days. The delivered cost of the major components of machinery and equipment, multiplied by a factor of from 2.2 to as much as 4.1, will serve the purpose of a flash estimate. Information is generally accumulated from local machinery suppliers, with no attempt made to get actual quotations on items. Auxiliary buildings, railroad spurs, power lines, water systems, and the numerous plant auxiliaries are computed as individual items, based on the experience of the estimators, and added to the factored price of the plant itself.

For a "Shotgun" Estimate:

(2.2 to 4.1) x Delivered cost of major components of machinery and equipment
Accuracy — 20 pct plus or minus

The true estimator has a feel or intuition in establishing the multiplying factor which is gained only through long experience and thoroughness in attempting to cover all possible contingencies. For a normal installation without too many problems of location, weather, etc., a factor of 2.75 times the delivered cost of the machinery and equipment can be a fair estimate of plant construction. A complicated plant with an abnormal amount of piping, electrical installation, controls, and instrumentation, may increase this factor to as much as 3.5. This type of estimate is at best a shotgun or an educated guess. The degree of accuracy is probably plus or minus 20 pct, depending upon the ability of the man who puts down the final factor.

Assume that management has been agreeably surprised by the shotgun estimate, and appropriates funds to further define the figures. Preliminary layouts indicate size of buildings, and a more precise equipment list is prepared. Accumulation of data usually requires a much longer period of time than for the shotgun estimate, and factors can be applied which may make the estimate a little more sound.

To the price of the machines delivered to the job site, 20 to 30 pct is added, considered as piping, insulation, and instrumentation. Add to this total 25 pct for erection and installation. Then calculate the buildings, foundations, and the electrical power as separate units based on square footages and unit costs, and add this to the first three items. Add an expense item of approximately 8 pct, and after the expense item, a field expense item of 10 pct. At this point addition for contingencies, estimating, and start-up is made, 15 pct of this subtotal. Provisions for contractor's overhead and profit vary with each individual negotiation. This intermediate grade estimate can be used to establish the contractural relationship between the engineers and the contractor, and define the scope of their work for purposes of contracting, and can then go to management for negotiation. This estimate is accurate to plus or minus 10 pct.

For an Intermediate Estimate:

To the price of machinery delivered to the job site—
1—Add 20 to 30 pct — Piping, Instrumentation, Etc.
2—To (1) add 25 pct — Erection and Installation.
3—Calculate Buildings, Foundations, and Electric Power on unit costs and add to (2).
4—Add 5 pct Expense. 5—Add 10 pct — Field Expense.
6—Total (3), (4), and (5) and add 15 pct — Contingencies, Estimating, and Start-Up.
Accuracy — 10 pct plus or minus.

The final estimate is detailed from the completed drawings and specifications. At this point it is imperative that a check list be prepared. The plant is completely engineered and every cost item can be established. Labor conditions and efficiency in a particular area are known and all factors are incorporated into one final picture which should be accurate.

An estimate in the early stages of a project is often based on knowledge of what a similar plant cost. Adjustments are made for differences due to geographic location and time of erection. The *Engineering News-Record* indices of construction costs are excellent for making these adjustments rapidly. Underestimating or overestimating by large percentages are both bad, the most likely error, however, is underestimating, particularly in preliminary estimates. Here it is due to inherent scant information, and unwillingness to apply large contingencies to compensate. Again, many times the low estimate is encouraged because of preconceived ideas as to cost, or for fear of scaring prospective financing away from the project.

These troubles do not arise often on estimates prepared by contractors for use as bids. By this time plans are complete, all details are known, and accurate take-offs are possible. Secondly, just a few low estimates can put the contractor out of business. Where competition is available, overestimating is competently penalized.

Estimating engineering costs is usually comparatively simple. Sometimes this is done by relating manhours to estimated expenditure. An example of this is to take 30 to 50 man-hours per $1000 of

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plant investment, or to take 4½ to 12 pct of total job cost, depending upon project complexity.

Estimated engineering cost is usually the one factor that inexperienced management argues. An organization is better prepared to make mistakes on paper that can be corrected by an eraser at \$10 to \$15 an hr compared to having a large crew shut down in the field. However, in seven cases out of ten, the management or client will take exception to the estimated amount of engineering.

In closing, the author would like to state that the chemical and petroleum industries are far ahead of the mining industry in methods of estimating as well as in methods of engineering. During 1951 a symposium on cost estimation was held under the auspices of the American Chemical Society and several papers were published from the symposium in *Industrial & Engineering Chemistry*.

Chairman Michaelson: As an effort to prove to you that estimating is, or should be, a scientific form of engineering I thought that some of the group would be interested in what the *Engineering News-Record* Index is.

If a plant cost "X" dollars fourteen years ago, how much will it cost today? The *ENR* Index is made up of cost of structural steel, cement, lumber, and common labor. If we take January 1948 as a base period; in January 1949 prices had risen 8.9 pct, January 1950 was 9.7 pct over 1948, and 1951 was 20.44 pct over 1948. In 1952 the Index was 24 pct more than 1948, and the last round, 1953, made it almost 33 pct more than 1948. That Index is based roughly in dollar value on 61 pct labor. Speaking from hearsay, some of the larger construction companies have developed their own indices which do not carry labor as 60 or 61 pct. A great deal is said about increased productivity of labor, but in construction it seems to be different and the average index of several larger corporations is 90 ot 95 pct of the total index. This means that labor has become less productive on construction work.

I am just a plant engineer, but I know that often estimates are made in a hurry. Sometimes when in the cafeteria somebody will ask "How much will a new plant cost?—Oh, just a rough figure, it doesn't have to be very close." Often circumstances are such that we end up building plants for that "bowl of soup" figure. . . . I wonder if some of the contractors might care to tell us what happens when a "bowl of soup" figure is what you actually build a plant for?

W. L. Howes: *(Western Machinery Co.,* Hibbing, Minn.) The sad part of these shotgun estimates is the permanency. I have been maneuvered into that position sometimes when asked for "just a guess." The next thing that happens is that it grows into a firm commitment. Oftentimes it calls for a lot of maneuvering to get out of that firm commitment. It is quite true that given an equipment list one can arrive at a reasonable figure. It takes a lot of practice and a few crystal balls. But, again I caution people in the matter of "guesstimates."

B. A. Reak: *(Roberts & Shaffer,* Hibbing, Minn.) It takes considerable time to arrive at a lump sum bid. In method, the customer gives the flowsheet and we work out a design drawing and figure every piece of equipment. From that we get a lump sum figure. Not only that, but we write specifications so the customer knows what he's getting. Sometimes we don't get the job after spending a month or six weeks getting the bid ready. . . . One method of estimating engineering hours is to use the estimate and figure how many drawings it is going to take to do the job. It costs money to design a plant and when you give a figure the contractor isn't making a fortune.

L. C. Raymond: *(Ford, Bacon & Davis,* New York City.) We find that operators make preliminary estimates, but sometimes they have in the back of their minds costs from before 1948. And I think the thing that disturbed a lot of estimates in recent years has been the change to shorter working hours. You make a detailed estimate, and then find out that the mine wants to go to a five-day week instead of a six-day week. It is amazing the change this makes in surface features, storage facilities, etc.

Marc Lintz: *(Consulting Engineer,* San Francisco.) When I started engineering work some contractor would tell me how much something was going to cost — a quick estimate. I would make a detailed estimate, and the original figure was almost as close. I learned to respect those estimates, if they were mature judgments, and finally used them myself.

Chairman Michaelson: Based on quite a few million dollars of construction I am inclined to agree wholeheartedly. We also do the kind of estimating described in the paper, naturally. But I wonder in today's construction market, if the detailed estimate is worthwhile.

H. V. Hughes: We are often accused of taking copious notes and getting dubious results, but people building plants have their own accounting departments.

Chairman Michaelson: Probably because of experience we have a separate accounting department. As far as the extras go, that is true. We clip ourselves, too, because of flowsheet changes. I still stick with the statement that in today's market the value of extreme details in estimating is questionable.

J. D. Grothe: (*The Dorr Co.*) Rely on the fact that if you have many items in detailed estimates, your errors will compensate and that is where the greater accuracy of the estimate may lie, rather than in the detail work.

R. E. Crockett: (*Consulting Engineer,* New York City.) In an engineering organization I was trained in the type of detailed estimate that has been described. I am now in an advisory capacity for six or eight companies and asked by clients what a job will cost and have to guess. Well, that guesstimate is based on my original detailed training, plus accumulated experience.

Question from the floor: How much variation do you get in detailed estimates percentagewise? **H. V. Hughes:** It is a lump sum bid and should not vary, because then you lose money. If you are talking, it is one thing; if you are testifying, it is another. On a lump sum you are testifying.

Editor's Note: A paper presented as part of the symposium by O. W. Walvoord, which illustrated problems met in the design of a specific modern milling plant, will be published in a future issue of MINING ENGINEERING.

Helicopters for Exploration?

The Problem:

There are no roads and mapping has been inadequate. There isn't a clearing large enough to accommodate a conventional airplane. Entry would consume months, yet the area must be investigated. Sustained communication is impossible except by radio. Cost in time and money for exploration would be exorbitant. . . . .

by M. A. Matzkin,
News Editor

BUT a growing group of men with the know-how may supply the answer to geologists and other mineral industry men who are faced with this kind of problem in the future. These men are the pioneers of the helicopter. Military prototypes proved invaluable both on land and over the sea during the Korean War. Many GI's owe their lives to the helicopter and the men who fly them. Because they can land on the proverbial dime, helicopters have access to areas that stymie jeeps, and even men on foot.

What does this mean to the mining man? Can the helicopter help him? The answer varies from person to person and from company to company. Some foresee a great future for the helicopter while others point to high cost and comparatively small capacity.

One of the men who believe in the helicopter is Neil Campbell, chairman of the Field and Applied Geology Committee of the Canadian Institute of Mining and Metallurgy. He points to advances in the techniques of field geology made possible by the helicopter. The machine has permitted maximum use of the short northern Canadian summer. Mr. Campbell also emphasizes the flexibility of application and uniformity of coverage by helicopters.

The late Sir Cyril S. Fox offered evidence of the value of the helicopter several years ago in his description of the geological exploration of the Wadi el Raiyan basin in Africa. He used a Hiller 360 helicopter. Complete details appeared in the *Mining Journal*, (London) October 1951.

On the basis of comparative costs the helicopter comes out second best to conventional fixed wing aircraft. A machine large enough for an adequate freight carrying job — the type needed for bush exploration — might cost $140,000. Gas consumption according to several exploration executives is much too high. Yet some of the company spokesmen felt that there may be a definite place for the helicopter when the cost per machine is lowered.

One company, currently involved in Ontario mineral development, states that its geological people feel that conventional aircraft are preferable. The helicopter is too small, too expensive, and will not carry large parties. Groups transported may reach as many as 50 persons. They prefer the Husky and the Norsemen type aircraft. However, the geologists are of the opinion that for short hops to inaccessible places the helicopter has its uses.

A company spokesman for Falconbridge Nickel Mines Ltd. suggests that his organization is not unaware of the potential:

"We have not so far made use of helicopters in our exploration. For large aerial coverage by magnetometer survey we have relied on standard aircraft . . . . but [we] are quite appreciative of their usefulness, particularly in topographical and geological reconnaissance."

One of the strongest cases for the helicopter is presented by C. S. Lord, geologist, Regional Geology Div., Geological Survey of Canada. The helicopter had a real chance to show what it could do in Operation Keewatin. Mr. Lord points out that a "radically new field technique" was needed if advances in geological reconnaissance were to be made beyond Yellowknife in the Canadian Shield. Mapping the area was a tough job. Canoes could be used only for a short time during the summer months. Parties were scattered over a wide area.

"Speed, accuracy and economy were necessary," and the helicopter appeared to offer the answer as early as 1946. Suitable aircraft were unavailable at that time.

For Operation Keewatin, Kenting Aviation Ltd., supplied two Hiller 360 helicopters, equipped with floats and extra fuel tanks. During a field season, lasting from May 12 to September 1, about 57,000 sq miles were mapped. The helicopters consumed almost 7200 gal of gasoline during that time. An airplane supplied the helicopters and was also employed for freighting camp and other necessities.

Helicopters were used only for geological observation and for a few short hops transporting geologists from ground traverses in areas where conventional planes could not land. The machines flew from 3 to 3½ hours on normal traverses, with some 110 sq miles mapped during each helicopter hour. Unfavorable weather grounded the machines some 21 pct of the time but quite probably regular aircraft would not have been able to fly under similar conditions.

One definite advantage of the helicopter was its ability to hover over outcrops, giving geologists an excellent opportunity to make reliable aerial determinations.

According to Mr. Lord, in an article appearing in the *Canadian Mining and Metallurgical Bulletin*,

April 1953, total cost of the operation was $206,-911.48 exclusive of salaries for Department of Mines and Technical Survey men. Helicopters were the most significant entry on the expense side of the ledger. Contracts stipulated that the helicopter contractor received about $163 per helicopter hour, or about $98,000 for approximately 600 helicopter hours. While the cost appears to give the helicopter no advantage over the fixed wing airplane, the machine accomplished a task that would have taken ground parties, mapping on a scale of 1 in. to 4 miles, some 25 years. It allowed the maximum use of personnel and gave uniform coverage, according to Mr. Lord.

Probably the armed forces have acquired more experience with helicopters than any other group. Col. Robert R. Robertson, Corps of Engineers, lauds the use of the helicopter in Alaskan mapping operation during 1952. In an article in the September-October 1952 issue of the *Military Engineer* he states that the helicopter increased production of the 30th Engineer Battalion by 500 pct. Small utility machines with payloads ranging from 350 to 600 lb were employed. Actual payload of a helicopter depends upon atmospheric conditions, altitude, and bulk of the material handled.

One item that has to be watched carefully is the balance of the load on the helicopter. As gas is consumed in flight, the center of gravity changes, often forcing the nose of the helicopter down.

Col. Robertson says, "The value of the helicopter in field survey can be positively stated. Until electronic equipment . . . . will yield the required accuracies many critical areas of the world can be accurately mapped only by use of the helicopter."

Charles B. Badgett, 1st Lt., Corps of Engineers, foresees helicopters with a load capacity of 20,000 lb in "Helicopter — New Tool of Engineers."

"Helicopters with detachable pods, equipped as shops, map reproduction units, small spare parts storage, and water supply units could be moved and spotted with great rapidity."

That kind of ability could have a lot of meaning to the mining man. Right now, the army man admits that the initial cost is big, and maintenance is a major problem.

Texas Gulf Sulphur Co. is using helicopters on a limited basis and with a certain amount of success.

Hudson Bay Mining & Smelting Co. is operating this Sikorsky S-55 in the White Horse area of the Yukon. It has a carrying capacity of 1800 to 2000 lb, or eight to ten persons, depending on distance flown. The machine has been in operation all summer on survey work. There is some thought being given to its use for hauling men and supplies for drilling operations. Sikorsky officials state that a concentrated sales effort is being made in the mining field.

The company employs the machine for transportation to its many Gulf area operations, lacking landing facilities for fixed wing aircraft.

Seismograph crews working in the Louisiana marshlands have found the helicopter one of their best friends. A breakdown in equipment once meant the end of the day's work. Now, helicopters scoot in and out of the working area. Spare parts are available on short notice. In some cases crews have experienced as much as one-third increase in production. Primarily, the helicopter has served as an aerial taxi. However, it can do other important jobs such as dragging a surveying chain across tough terrain and picking up seismograph records for delivery to the computer at the quarter boat.

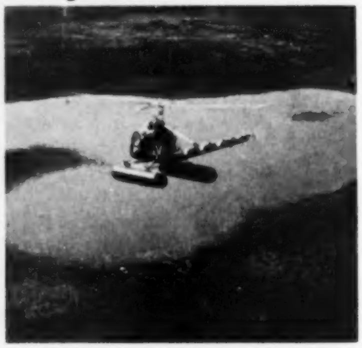
One of the outstanding helicopter performances has been turned in by Carl Agar. Aluminum Company of Canada solved several near stymies by employing Okanagan Helicopters Ltd. Agar, vice presi-

A skid equipped Bell 47D-1 working for Alcan hovers over a landing site before discharging its load. Smaller machines like this one have been used successfully on the Kitimat project.

Helicopters Solve Problems of Snow, Water, Tight Landing Areas

Postage stamp sized airports are no problem. This 14-ft square heliport is more than adequate for the Bell helicopter. The port is located at one of the mountain camps set up by Alcan at Kemano, B. C.

Take off and landing from the glacier at the head of Kildala Pass is one of the things Okanagan Helicopters Ltd. took in stride. Here the 'copter comes in for a cargo drop at Camp No. 10.



A pontoon equipped helicopter can land on water as well as land. Thus, it becomes especially adaptable to a region filled with small lakes, but where some land operations must be carried out.

dent of the airline, was in on some of the original exploration which inaugurated the Kitimat project. One of the most significant jobs done by the helicopter involved construction of a $350,000 aerial tramway from the base camp at Mount Dubose up the mountain side. Most of the equipment for the tramway went up the slope via helicopter.

Agar has been supplying some 21 camps over a 150 mile area using Bell and Sikorsky machines. Incidentally, Agar and his men flew in most of the material needed to set up these camps. In fact, even the base camp at Mount Dubose owes its existence to helicopters. Men on foot failed in attempts to set up the camp and it was Agar who finally turned the trick.

Two aluminum towers used for power transmission from Mount Dubose were air-lifted by helicopters to the top of Kildala Pass, topmost spot on the projected power line route. Agar picked up and deposited instruments, men, and supplies at the construction site, landing on postage stamp sized areas.

Since then, according to one report, Agar has been helping to build roads, deliver supplies, and make other jobs easier with the helicopter.

During Operation Keewatin helicopters ordinarily were equipped with floats, but still were able to drop geologists directly on the required outcrops. The machine is a Hiller 360 carrying pilot and one geologist. Helicopter unserviceability was only 3½ pct of the available field season during the operation.

The obvious advantage of the helicopter in airborne magnetometer work is its ability to get down close to the ground. Fixed wing airplanes must fly high enough for safety and anomalies recorded can be keyed to ground position within limits no closer than several hundred feet. One of the pioneers in the use of the helicopter in this type of work has been Hans Lundberg and his associates. In the September 1946 issue of *Mining & Metallurgy* it was reported that recordings were made at altitudes of 150 to 250 ft. Helicopters permitted detailed study of any point of specific interest. Surveys were conducted in the Sudbury area during 1946. Helicopter-borne magnetometer results exceeded expectations because a two-hour flight yielded results originally obtained from 108 days of conventional ground survey.

In a letter to the editors in the May 1950 MINING ENGINEERING, G. T. Warren and W. J. MacKenzie stated: "During the course of recent prospecting for commercial minerals in southern British Columbia, we have discovered that the helicopter is a very useful method of accelerating mineral exploration, assisting materially in the initial development of potential mines, and facilitating topographical and geological surveys."

The authors claim that useful preliminary surface examinations can be performed with the helicopter because of its ability to fly close to mountainsides at low speed, and hover over talus slopes and stream beds.

However, as others have pointed out, flying a helicopter in mountainous country is quite a job and not something for the amateur. The pilot must know the hills, air currents to be expected, and be prepared to fight off the effects of treacherous winds. In competent hands the helicopter is as safe as any other means of mountain transportation.

It would seem that the age of the helicopter as far as mining is concerned is sometime to come. Payloads are too small in the opinion of a great many authorities and the cost is high. Yet, many have found it to be a useful tool and see a growing place for it in the industry. It can't carry the load that the Norseman can, and it can be used only in special cases today, but the era of the *whirlybird* may not be too long in coming.

Middle East's Only Asbestos Mine Operated On Island of Cyprus

A typical Greek miner at the Tunnel Asbestos Cement Co. mine at Amiandos.

THE only asbestos mine in the Middle East is on the Island of Cyprus, high in the Trodos Mountain village of Amiandos. The mine is owned by the Tunnel Asbestos Cement Co. Annual output is some 18,000 tons of short fibre in the 4-T, 4-Z, and 5-R grades.

Since the company was first registered in 1907 at the island's capital, Nicosia, more than 200,000 tons of asbestos have been mined via the open pit method. The asbestos goes to Japan, Germany, Sweden, Egypt, Israel, Argentina, Siam, Spain, and other countries. Ore is dislodged using electric drills and blasting techniques and sun-dried before processing. Eight mills working on a 24 hour schedule process ore during summer months.

The village itself is company owned. Some 2300 men and 500 women are employed in the mine and mills. One of the reasons for the number of women workers is the need for a marriage dowry. Pay for women is about $2.00 per day and men average about $2.50.

While Cyprus is a British Colony, the 80 pct Greek population constantly discusses union with "mother Greece." Ninety pct of the mine workers are of Greek origin, with a sprinkling of Turks and Armenians.

Many women are employed in the mine. Women earn about 50c per day less than men.

One of the company mills is located at the end of the village street.

The mine and mills are entrenched in Trodos Mountains. This is the only asbestos mine in the Middle East.

Labor Legislation —— As It Affects the Engineer

According to the June 27 issue of *Business Week*, Rep. Kearns (R., Pa.) introduced a bill in Congress along lines suggested by the National Society of Professional Engineers, to amend the Taft-Hartley Act to permit scientific, technical, and professional groups to discuss wages, hours, and working conditions with management—with the protection and support of the federal law. The article reports that the bill is opposed by a loosely knit group of professional and technical unions, the Engineers & Scientists of America. This group wants Taft-Hartley amended to make it an unfair labor practice for employers to discuss collective bargaining matters with professional organizations.

One interested member of AIME reported to the Institute that he felt that he had the right to discuss wages, hours, and working conditions on his own initiative with the company he worked for. He was opposed to any group, professional or union, usurping this right. He also felt that if engineers are ever to obtain professional status, it will be on the basis of their individual ability and attainment rather than on bargaining agreements that might be pressured by a group.

To clarify this matter, AIME's representative on the Engineers Joint Council Labor Panel prepared the following outline of the national Labor Act as it affects professional engineers and the part the professional societies have played in modifying it in the interest of professional employees.

As it now stands, the Taft-Hartley Act allows professional employees: 1—To form their own collective bargaining groups; 2—to unite with some other group of their own choice; or 3—to reject collective bargaining entirely. It provides that the Labor Relations Board shall not decide that any unit is appropriate for purposes of collective bargaining if it includes both professional and nonprofessional employees unless a majority of such professional employees vote for inclusion in such a unit.

Under the original Wagner Act, no distinction was made between professional (engineers, doctors, etc.) and other employees (carpenters, bricklayers, plumbers, etc.). A majority vote within a plant to establish a collective bargaining agency brought under the agency all employees, engineers included, whether they liked it or not, and they were governed by any instrument of bargaining executed between that agency and their employer. There was no recognized definition of a professional employee, which made it next to impossible for professional employees to form their own organization if they wanted to.

The ASCE was first to be aroused to the necessity for some action to protect the interests of a large group within their membership who came under the Act. It supported various means for maintaining the integrity of the engineer in matters of collective bargaining, and finally earned the sympathetic cooperation of other member societies of Engineers Joint Council, with the result that EJC appointed a Labor Legislation Panel in which ASEE and NSPE joined. As a result of the action of this group, following the election of a Republican Congress in 1946, a definition of a professional employee and the stipulation concerning his rights in matters of collective bargaining, as listed above, were written into the new Taft-Hartley Act.

Therefore, all engineers not bound to any organization which they have voted to recognize as their representative in matters of collective bargaining have the right to discuss wages, hours, and working conditions on their own initiative with the company they work for.

The NSPE and EJC are concerned about an interpretation of the Taft-Hartley Act as it now stands and as it affects technical associations. The matters in question bear on what has been termed Freedom of Association. Throughout the country there are groups of engineers banded together to discuss matters which logically come under the head of engineering and which may involve conditions of working and wages, but purely from an engineering viewpoint. The groups may be composed of representatives of management, possibly executives, as well as rank and file engineering employees. Now the present Labor Act defines a *labor organization* as "any organization of any kind, or any agency or employees' representation committee or plan, in which employees participate and which exists for the purpose, in whole or in part, of dealing with employers concerning grievances, labor disputes, wages, rates of pay, hours of employment, or conditions of work." Under this definition the technical groups described could be called *labor organizations* and, as such, be subject to the regulation of the Act concerning matters of "unfair labor practices" and the like. If it be construed that any professional society exists for the purpose of dealing in part, with rates of pay, hours of employment, or conditions of work, it could be branded as a *labor organization* under the definition in the Labor Act.

The protection afforded such technical organizations, however, appears in the "Freedom of Speech" portion of the Act which says: "The expression of any view, argument, or opinion, or the dissemination thereof, whether in written, printed, graphic or visual form, shall not constitute or be evidence of an unfair practice under any of the provisions of this Act, if such expression contains no threat of reprisal or force or promise of benefit." Thus while under the Act the society could be branded as a *labor organization* by definition, it would be difficult to find it guilty of unfair labor practice under the "Freedom of Speech" provision.

The interests of the engineer cannot be taken care of by ignoring labor legislation. The AIME is not engaging in collective bargaining arrangements. Through the Labor Panel of EJC it is trying to preserve the right of the individual engineer, who has not elected to join a collective bargaining group, to discuss wages, hours, and working conditions on his own initiative with his company. Engineering groups have no desire to discuss matters of labor from the viewpoint of collective bargaining—only as a purely engineering concern.

The interest of AIME and EJC in matters of labor legislature is that which all engineers must have to preserve what independence of action the individual engineer desires.

Sulphur Recovery From Low-Grade Surface Deposits

by Thomas P. Forbath

THE sudden realization that known sulphur reserves amenable to mining by the Frasch hot water process are nearing exhaustion focused attention on widely scattered surface deposits throughout the world. These deposits are not necessarily of lower sulphur content than ores located underneath Louisiana or Texas salt domes which usually average about 30 pct sulphur disseminated in limestone matrix. Their near surface occurrence, however, renders exploitation by the Frasch process impossible. As is well known, the Frasch process depends on the presence of 500 to 1000 ft of overburden and cap rock above the sulphur deposits to permit melting underground sulphur in place by diffusing hot water under pressures of 200 to 600 psig in the formation and raising the molten sulphur to surface by air lift. This process renders possible the production of pure sulphur which is 99.5 pct pure without any subsequent treatment.

Surface deposits contain sulphur in the same range of concentrations as the salt dome deposits, i.e., from 10 to 50 pct sulphur, associated with various gangue materials such as silica, limestone, and gypsum. The principal distinction, then, does not lie in the percentage of sulphur contained in the ore, but in the geological nature of the deposit.

A recent study[1] of the world sulphur supply situation estimated 1950 sulphur production in the free world countries at 5.6 million long tons, of which the United States produced 5.2 million tons, or 93 pct of the total. While America's domestic needs alone would have been covered by national production, about 1.4 million tons were exported during the same year. Despite all the steps which are being taken to restrict use of elemental sulphur and to force the fullest possible development of alternate sulphur sources here and abroad, the deficit in elemental sulphur production will probably increase with time. As a result of intensive prospecting for oil throughout the Gulf Coast area discovery of significant new salt domes is held unlikely. With the growing scarcity of sulphur and what appears to be an inevitable rise in price, recovery from deposits not amenable to Frasch-process mining assumes greater economic importance.

Untapped Reserves

The most important deposits in this category are located in Sicily, where elemental sulphur occurs in Miocene limestone and gypsum formation. Sulphur content of these ores ranges from 12 to 50 pct with an estimated average of 26 pct. Although quantitative estimate of these reserves is not available it is held that they exceed 50 million tons of sulphur. Similar deposits occur also on the mainland which contribute about one-third of Italy's total current annual production of 230,000 tons, but these are known to be nearing exhaustion.

Significant surface deposits of volcanic origin are located in South America, Japan and western United States, silica being characteristic gangue constituent. The largest of these deposits are in South America. More than 100 extend over a zone 3000 miles long, paralleling the west coast of South America. Total sulphur content of these deposits has been estimated to be as high as 100 million tons. The main islands of Japan also possess at least 40 known volcanic sulphur deposits with probable reserves of 25 to 50 million tons.[2] Prospected reserves in western United States might amount to 2 million long tons, principal deposits being located in the northwestern part of Wyoming, southern Utah, and eastern California.

Volcanic deposits of lesser importance are found around the Mediterranean, in Turkey and Greece, and in Africa, Egypt, Abyssinia, and Somaliland.

Beneficiation Methods

Different methods of beneficiation have been used in these various locations. In Italy the Calcarone kiln and Gill regenerative furnaces are used exclusively. Both utilize heat liberated by burning part of the sulphur in the ore to liquify or vaporize the remaining sulphur, which is recovered by solidification or condensation. The Calcarone kiln is of conical shape, generally 35 ft in diam at base and 18 ft high. A kiln of 25,000 cu ft capacity burns for about two months and yields about 200 tons of sulphur. The Gill furnace consists of a series of chambers with domed roofs. Sulphur is burned and melted in one chamber at a time and the hot combustion gases are used to preheat the ore charge in the subsequent cell. These furnaces operate on a cycle of 4 to 8 days. The recovery yield of both systems is about 65 pct. Sulphur losses amount to 25 pct through the combustion to sulphur dioxide; about 10 pct is retained in discarded calcines. Ores containing less than 20 pct are not considered suitable as furnace feed.

These methods are not only wasteful because of the low recovery obtained, but represent a serious atmospheric pollution problem. Sulphur produced ranges from 96 to 99 pct purity and thus does not match Texas or Louisiana sulphur. Owing to the present shortage, sulphur in the Middle East sells

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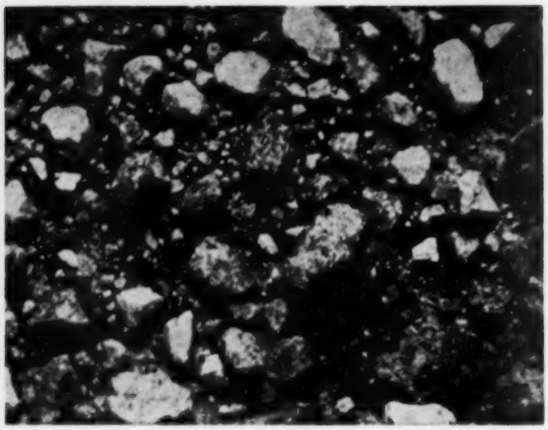
The Transactions papers appearing in Mining Engineering during 1953 will subsequently appear in AIME Transactions Vol. 196 and may be permanently referenced to that volume.



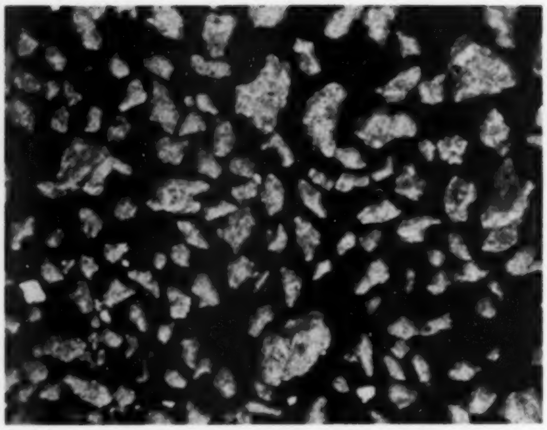


Fig. 1 left)—Sulphur ore from Colombia, S. A. Even fine grinding does not liberate all the sulphur, which is thoroughly disseminated in the gangue. Fig. 2 (right)—Oversize particles of sulphur formed by coalescence of sulphur droplets after treatment in gangue separator. Both views X3.

for as high as $200 per ton. In spite of this, the Italian sulphur industry is still beset with economic difficulties. These can be ascribed to high mining costs, averaging about $8 per ton of ore, as well as to high recovery costs.

In South America the batch autoclaving method is widely used. In this process sulphur ore is crushed to about 2-in. size and a batch of approximately 1.5 tons is charged to cast iron autoclaves 4 ft in diam and 7 ft high containing a grate with ¼-in. openings on which the charge rests. After charging, the autoclaves are bolted and live steam of 60-lb pressure is admitted for a heating period of 1 hr. At the end of this period, a valve at the bottom of the clave is opened and the molten sulphur which passed through the grate is drawn off into concrete pits where it solidifies. The autoclave is then opened and the leached ore removed and discarded. The complete cycle takes about 2 hr. Since the steamed ore still retains 20 to 30 pct sulphur, recovery yields are always very low. Although the purity of sulphur produced by this method is acceptable, production costs are excessive because of low recovery and high utility and labor costs caused by the batch-type operation. Even at the present time, when sulphur sells at $100 and higher at South American ports, the autoclaving method offers a narrow margin of profit at best.

Several attempts have been reported on the application of straightforward froth flotation methods to recovery of sulphur from surface ores.[3,4] While sulphur is considered to be a *natural floater*, it was usually found that production of a high purity concentrate with high recovery yields was not possible. This difficulty is largely due to the fact that in most ores sulphur is very finely disseminated within the gangue and even after fine grinding (100 pct through 200 mesh) microscopic examination shows a large proportion of sulphur and gangue particles cemented together. Using more extreme grinding renders the flotation step more involved. Finely ground sulphur flocculates very intensely and occludes particles of gangue. These occlusions are hard to break down.

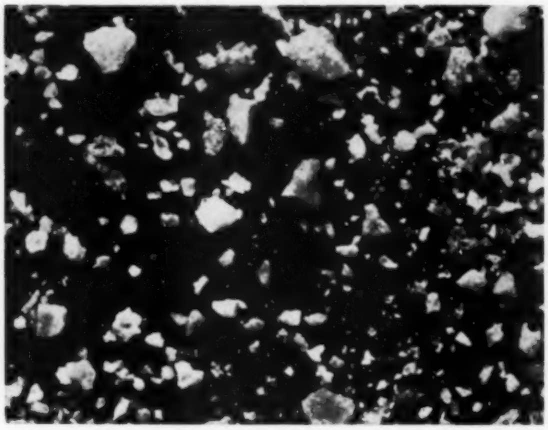
Processes employing solvent extraction to recover elemental sulphur from ores have also been proposed and described in literature.[5] The drawback of these processes usually lies in high vapor pressure, inflammability, and cost of solvents involved. While use of carbon bisulphide, for example, is technically feasible at sea level, CS_2 would behave as a gas at the altitude of most of the South American deposits, and at the temperatures involved in a solvent extraction process.

These and other methods were used at one time or another at nearly all the known sulphur deposits throughout the world. None proved competitive with Gulf Coast sulphur.

The Chemico Process

For over 10 years, Chemical Construction Corp. has received inquiries from foreign countries regarding concentration of sulphur ores, and about 5 years ago decided to attempt development of a process which would yield high purity sulphur at an economical cost. The first ores tested under this program originated from Colombia, S. A., Egypt, and Milos Island, Greece. On each of these ores, recovery by flotation alone did not prove feasible, even with fine grinding to—200 mesh. The idea arose of separating sulphur from gangue by melting prior to flotation. Use of high boiling liquids at atmospheric pressure was first considered. Experiments in which ground sulphur ore was suspended in 98 pct sulphuric acid and heated above 240°F, i.e., the melting point of sulphur, under agitation, showed the way to further developments. It was observed that in relatively short times all the sulphur could be melted and that the molten sulphur droplets tended to coalesce and agglomerate. Also noted was the fact that following such thermal treatment, subsequent separation by flotation was relatively easy.

The idea of handling large amounts of hot sulphuric acid in commercial installations was not very attractive; consequently in the next step of the development program it was attempted to melt sulphur away from gangue by suspending the ground sulphur ore in water and heating the slurry under a pressure of 35 psig above the melting point of sulphur, while maintaining it in a state of agitation. It was again recorded that in relatively short periods, from 15 to 60 sec, all the sulphur could be physically separated from the gangue. At the end of the heating period cold water was injected into the agitated mixture and the temperature reduced to 185°F, freezing all liquid sulphur particles. Subsequent recovery of sulphur by froth flotation was again found to be relatively simple. These experiments provided the basis for the present Chemico sulphur recovery process, since thermal treatment in the gangue separator is considered the key to its suc-



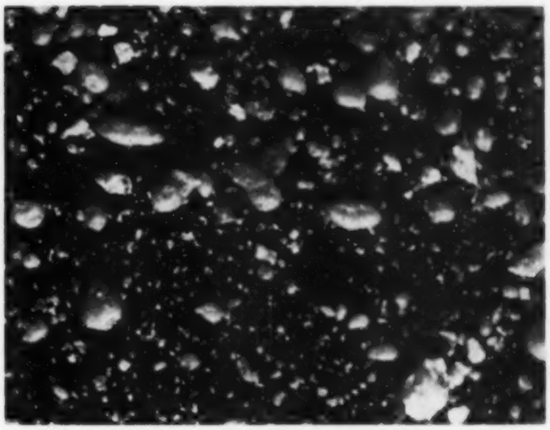


Fig. 3 (left)—Rougher flotation follows thermal treatment, yields this tailing containing 2 to 3 pct S. Fig. 4 (right)—Cleaner flotation cells produce a 90 to 95 pct S concentrate, and a middling which is recycled to rougher flotation. X15.

cessful application to a wide variety of highly refractory ores.[6]

To adapt the thermal treatment step to a continuous process a coil-type reactor was developed. Ground slurry was pumped through a pipe coil at velocity sufficient to provide turbulent flow conditions. Live steam of 35 psig pressure was injected into the coil, which was kept under a pressure of about 30 lb. The coil was made long enough to provide retention time of about 1½ min. At the tail end of the coil cold water was injected to quench molten sulphur particles and the cooled slurry was then discharged through a pressure relief valve.

It was found that the thermal treatment step obviated the necessity of fine grinding usually required for satisfactory separation by straight flotation alone. This means a considerable saving in power consumption, since power required to reduce a given amount of sulphur ore feed from 12-in. size to—200 mesh is roughly three times that required to go from 12 in. to—20 mesh.

Process Details

The Chemico process developed on the basis of the above considerations can be described as follows. Ore is crushed in primary and secondary crushers about ¾-in. The crushed ore, Fig. 1, is then fed to a pebble mill which grinds it —28 mesh. Since dry grinding of sulphur-bearing ores is inherently dangerous because of fire and explosion hazards, wet grinding is employed. Pulp concentration in the pebble mill is kept about 50 to 60 pct solids. The pebble mill is operated in a continuous closed circuit with a 28-mesh screen. The grinding step invariably causes disintegration of some 50 pct or more of the sulphur into particles of 1 to 5 micron size. The ground ore is next suspended in water to form a slurry of about 30 pct solids content. This slurry is then fed by means of high-head diaphragm pumps to a coil-type gangue separator. Live steam is injected into the gangue separator which is maintained under a pressure of 60 psig, heating the slurry to 250° to 275°F and melting the sulphur. The high surface tension of the molten sulphur prevents it from wetting the gangue particles, and collisions caused by eddy currents due to turbulent flow coalesce the small sulphur droplets, which agglomerate into globules up to 2 mm diam. Before leaving the gangue separator, the slurry is quenched to about 190°F by the injection of cold water. Slurry leaves the gangue separator through a pinch-type pressure let-down valve and passes on to a 20-mesh vibrating screen. The agglomerated particles containing from 96 to 98 pct sulphur are retained on the screen and go directly to a sulphur melting pit. Usually about one-third of the sulphur is recovered in the form of these oversize particles, Fig. 2.

The balance of the slurry containing all gangue and non-agglomerated sulphur passes to a conditioning tank. In this tank flotation chemicals are added and pulp density is adjusted to about 20 pct solids. In most instances pine oil, fuel oil, or Aerofloat flotation reagents are used. The quantities required correspond to usual flotation practice, i.e., about 0.10 lb of frother and 0.05 to 0.10 lb of collector per ton of ore. From the conditioning tank

Fig. 5 (left)—Sulphur deposits at 12,000-ft altitude on the slope of Purace volcano, Colombia, S. A. Ore from this deposit is now being treated in the first full-scale plant to utilize the Chemico sulphur recovery process. Fig 6 (right)—Closeup of one of the workings supplying ore to the treatment plant.

Fig. 7 (left)—View of the commercial scale installation in Colombia, S. A. Fig. 8 (right)—Pebble mill in the Colombia plant. Mill has rubber lining that is further protected by silica blocks.

slurry is pumped to a bank of rougher flotation cells which produce a concentrate containing 70 pct sulphur. Tailing from the rougher cells, Fig. 3, containing 2 to 3 pct sulphur, is sent to a thickener which conserves both water and heat, since the clarified hot water is recycled to the process. Thickened tailing is discarded.

The product floated from the roughers passes to a bank of cleaner cells which produce a concentrate, Fig. 4, containing 90 to 95 pct sulphur. The middlings are recycled to the rougher cells through middling thickener which is required to maintain a water balance in the system. Concentrates are dewatered by a horizontal rotary vacuum filter and fed to the same sulphur melting pit into which the oversize sulphur was passed from the screen following the gangue separator. In this pit the sulphur is melted by means of steam coils and the molten sulphur is kept under continuous agitation to maintain impurities in suspension.

The molten sulphur is pumped to a pressure-type filter which yields more than 90 pct of the sulphur originally contained in the ore in the form of 99.5 pct pure sulphur as the final product of the process. The filter cake, which usually contains 40 to 50 pct sulphur, is recycled to the jaw crusher at the beginning of the process.

Engineering Problems

One of the principal engineering problems in developing this process was the selection of suitable construction materials. When sulphur ores of volcanic origin containing silica as the major gangue constituent are ground in water, the resultant slurry has a pH of about 2. Acidity is accounted for by oxidation of some of the sulphur by means of oxygen in the air or possibly oxygen dissolved in the water. The resulting sulphur dioxide forms sulphurous acid, some of which is oxidized to sulphuric acid. While the concentration of sulphuric acid in the slurry does not exceed a few tenths of 1 pct, the slurry is extremely corrosive. Taking into account the fact that ground silica is an extremely abrasive material it becomes evident that the handling of sulphur ore slurries represents a serious corrosion and abrasion hazard. It is known that the failure of several past attempts to recover sulphur from surface ores by means of flotation or autoclaving techniques can be attributed to the lack of corrosion and abrasion-resistant machinery.

Equipment used in the Chemico process is selected with due respect to the above considerations. The pebble mill is lined with rubber which in turn is protected by a lining of silica blocks. Silica pebbles are used as the grinding medium. The slurry mixing and conditioning tanks are lined with lead and acid-proof brick. For pumping the slurry, rubber-lined diaphragm pumps have been selected, after the testing of a wide variety of alloys in an attempt to find a suitable centrifugal pump. The entire piping, including the gangue separator, is built of No. 316 stainless steel, as are the vibrating screens, the horizontal vacuum filter, and the finishing sulphur filter. Flotation cells are built of California redwood with rubber-covered moving parts. To summarize, the only materials in contact with the hot acidic slurry are No. 316 stainless steel, rubber, lead, wood, and chemical stoneware.

To date more than 50 different ore samples from all over the world have been tested in the Chemico laboratory. The composition of the gangue and the nature of dissemination of the sulphur in the mineral influence considerably the ease of recovery. In general, ores of volcanic origin were found easiest to treat, but with slight modifications in the autoclaving and flotation technique the process was found to be successful on nearly every sample tested. Although some samples proved amenable to beneficiation by flotation alone with reasonable recovery yields, it was noted that samples from neighboring deposits exhibited a remarkable variety of behavior in this respect. After the thermal treatment step, however, samples which were highly re-

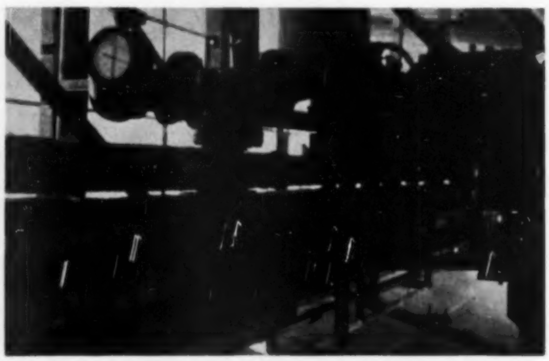


Fig. 9—Closeup of the gangue separator. Here slurry is heated to 250 to 275°F by 60-psig steam, then quenched to 190°F by injection of cold water. Sulphur droplets liberated from the gangue are partly recovered by screening (see Fig. 2), the remainder recovered by froth flotation (see Fig. 3).

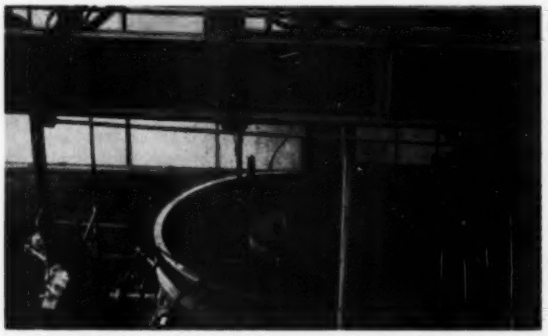


Fig. 10 (left)—Concentrate is dewatered on this horizontal vacuum filter before going to sulphur melting pit. Fig. 11 (right)—Pressure-type filter receives sulphur from melted pit, yields more than 90 pct of original sulphur content of ore in form of 99.5 pct pure final product. Filter cake recycles to crusher at beginning of process.

fractory to ordinary flotation methods yielded very satisfactory results.

Following the work done in the laboratory on small pilot plant scale where quantities of a few pounds were handled, the process was piloted in a plant designed to treat 1000 lb of ore per hr, i.e., 12 tons per day. This pilot plant incorporated all equipment required to carry out the process as outlined above in a continuous fashion. The pilot plant was operated for a 3-month period.

Some pilot plant equipment such as piping and the coil-type gangue separator were considered expendable and were made of ordinary carbon steel. On all these parts severe corrosion was experienced. A steel sand pump used to pump slurry to the flotation cells, for example, corroded and eroded to the point of uselessness within a 24-hr period. All the stainless steel equipment used did not show any signs of corrosion after 3 months operation.

Two principal types of ores both originating from California deposits were tested. Both contained silica as the chief gangue constituent, but they differed in their pyrite and soluble salt, principally $FeSO_4$, content, and this difference very markedly affected their behavior in the sulphur recovery treatment.

One ore assayed 33.7 free sulphur, 3.3 pct pyrite, and 6.2 pct soluble salts. This ore was amenable to beneficiation by the standard Chemico technique with a 95 pct sulphur recovery of a 93 pct sulphur concentrate, which after the finishing filter gave a final product of 99.8 pct purity.

The other ore assayed 25.3 pct sulphur, 6.3 pct pyrite, and 11.7 pct soluble salts. The standard flow scheme required certain modifications to obtain satisfactory results with this ore. Since the technique developed in the pilot plant for satisfactory handling of this high pyrite ore was found applicable to other samples containing appreciable amounts of pyrite and sulphate, steps introduced into the process warrant further discussion. The principal modifications consisted of 1—a rougher flotation to double sulphur content of material fed to subsequent parts of the process and filtration and washing of this material to remove the soluble salts and 2—operation of the gangue separator in such a manner that almost no oversize sulphur was produced, so that all sulphur in the ore was recovered as a 90 pct flotation concentrate rather than a combination of oversize and concentrate. This procedure was required since it was found that pyrite particles are wetted by molten sulphur and therefore tend to contaminate the oversize sulphur. Photomicrographs of the interior of the oversize particles showed interspersed sulphur and pyrite crystals.

After detailed experimental study of the variables affecting agglomeration, a procedure was developed which almost entirely eliminated production of oversize particles in the autoclaving step in the gangue separator. This technique involved the use of an alkaline circuit and a short retention time as well as low temperature in the continuous autoclave. As a result of this investigation, flotation concentrates containing 93 pct sulphur with a 95 pct recovery were successfully produced from the high pyrite ore.

The first commercial installation utilizing the Chemico sulphur recovery process has been constructed in Colombia. This plant is located at the site of a very significant sulphur deposit on the slope of the Purace volcano at an altitude of 12,000 ft, Figs. 5, 6. The plant is designed to produce 33 metric tons of pure sulphur a day from 90 metric tons of ore containing an average of 40 pct sulphur, see Figs. 7-11. Equipment used and significant steps of the process are similar to those described under Process Details, p. 883.

Utility requirements per ton of sulphur produced in a plant of this size and handling a corresponding grade of sulphur ore are tabulated below:

Power, hr	65
Steam at 75 psig, saturated, lb	4450
Cooling water, gal	1900

Three operators and two helpers per shift are sufficient to man a plant up to 1000 tons of ore per day feed.

Detailed investment and operating cost studies for a plant producing 100 tons of sulphur per day, located either in Wyoming or California, show that such an operation would produce sulphur which could sell at a cost competitive with the present controlled price of $22 per ton.

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The Use of a Caved Block as an Ore Pass And Its Application to Open-Pit Mining

by H. Carroll Weed

By caving a block into the workings of its open pit and using the block as an ore transfer, the Inspiration Consolidated Copper Co. has solved a transportation and sizing problem, making possible a great expansion of open-pit methods as applied to Inspiration orebodies.

FROM 1915 to 1948 the entire production of Inspiration Consolidated Copper Co. was supplied from underground mining. The sole method used was block caving. Underground haulage and hoisting facilities were designed and geared to large-scale production.

Beginning in 1948 open-pit mining was substituted for block caving in a portion of Inspiration's Live Oak orebody. The idea proved so attractive that before the Live Oak pit had come into production, another pit on the Colorado orebody (now the Thornton pit) had been laid out and stripping started. It should be noted that these orebodies were not new and the entire program was one of changed methods of mining.

Since part of these orebodies had already been mined by block caving methods, it must be recognized that haulage levels had been established under the area or adjacent to it. The orebodies lay on the south side of Inspiration Ridge, while the shafts, crushers, and treatment plants are all on the north side of this ridge. The existing main crushing plant was designed to take a maximum of 12-in. material, and all ore was sized to this dimension by passing through grizzlies in the stopes.

In the original planning for open-pit work much thought was given to the transfer of ore to existing underground levels for haulage and hoisting from the regular shafts. If this could be done the necessity of crossing the ridge could be eliminated. Final decision called for cutting a road through the ridge, installation of a primary crusher at an elevation of 3968 ft on the north side of the ridge, and railroad haulage with existing facilities to the main coarse crusher. This decision was brought about by the difficulties of properly sizing and transferring to underground haulage large tonnages of coarse breaking oxide which lay on the upper benches of the proposed pit. Trucking over the ridge on a 7 pct grade would give a cheaper and more flexible operation in handling this material.

The benches in the pits are laid out at 50-ft intervals and designated according to elevation above sea level. The lowest bench in the original pit was laid out at 3500-ft elvation. All haulage roads are on a 7 pct grade and a vertical lift of 468 ft was considered about the maximum economically possible from a cost standpoint.

After 2 years' operation the advantages of pit mining, both from the standpoint of costs and flexibility of operation, became more and more apparent. Studies were then started to develop the extension of open-pit mining to more of the ore reserve than had been planned originally. The idea of transfer raises was again explored. Obviously, as elevation of the pit was lowered, shorter transfer raises would be needed to reach the main haulage levels. Transportation from the lower elevations to the rim of the pit by belt conveyor or skip hoisting was also considered. However, it was recognized that sizing would be required both for conveying and regulation of feed size to the main crusher plant. This would necessitate a fixed or portable crusher located somewhere in the pit. It was known that the sulphide ore at the lower elevations was softer and easier to break than the overlying oxide of the upper benches. This sulphide ore is very suitable for grizzly sizing.

Calculations indicated that for ore mined on 3650 elevation, costs for trucking to the primary crusher, crushing, and delivery by railroad to the coarse crusher would equal costs for dropping the same material to the 600 level, hauling underground, and hoisting directly to the coarse crusher bins, provided a suitable method of sizing and transferring could be developed. Above 3650 elevation costs would favor surface haulage; below they would favor underground haulage.

The idea of caving an underground block from the 600 haulage level directly into the bottom of the pit

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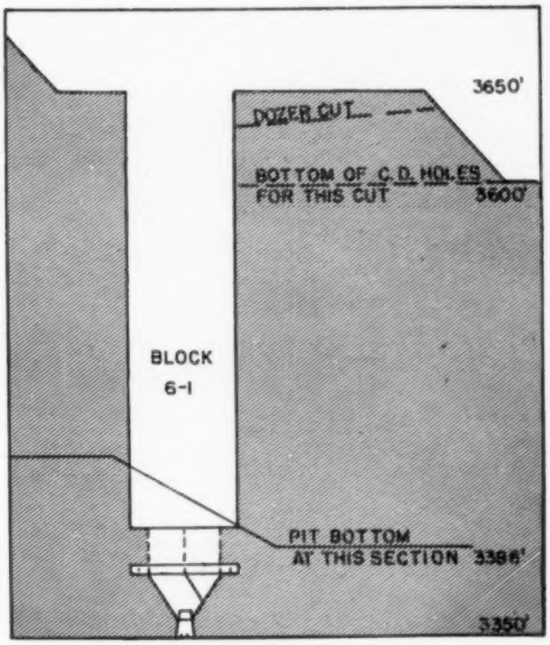

Fig. 1—Section through Block 6-1 and open pit.

and the continued use of this caved block as an ore transfer system developed from the studies made. Such a method provides for sizing, since all ore drawn must pass through the block grizzlies. The block provides plenty of storage capacity and a high rate of draw can be maintained, far beyond that obtainable from a regular transfer raise. Expensive upkeep of a timbered transfer raise is eliminated.

A suitable location was found in the Thornton pit, where the top of the ore was at approximately 3650 ft and where the 600 haulage level, elevation 3350, was reasonably close to the proposed block. Waste was stripped down to this elevation, and at the same time the block was developed on the 600 level. The block was in a granite porphyry which was somewhat altered. Such ground would stand well when timbered and cave freely when undercut, see Fig. 1.

In area the block is slightly smaller than normal, having but three grizzly lines of 33 1/3-ft spacing, three grizzlies in each line spaced on 20-ft centers and a total area of 60x100 ft. The raise system below the grizzlies consists of a single main raise from one side of the haulage drift and a main raise with a branch raise from the other side of the drift. Each of the six chutes in the haulage drift is equipped with an air-operated undercutting type of gate, Fig. 2, replacing the ordinary hand-operated arc type. With these gates, speed and ease of loading have been greatly increased and spillage held to a minimum.

Since it was anticipated that the block would be required to pass a very large tonnage, steel control sets above the grizzlies were used instead of timber sets, and experimental grizzly sets, some of steel and some of wood, were installed.

The block was undercut in the normal manner, but at 23 ft above the grizzly instead of the usual 18 ft. This undercut is at an elevation of 3409 ft above sea level, or 241 ft below the 3650 bench where the top of the ore was exposed. The block was undercut before stripping had been completed and was drawn very slowly. When less than 10 pct of the estimated tons in place had been drawn the stripping had been completed and the rate of draw was increased. The first hole appeared on surface when the stope was 18 pct drawn. This hole was within the projected block limits and covered about half the stope. The stope had been drawn down about 75 ft when it was observed that it had caved approximately full size to within 75 ft of surface, at which point a back was encountered which resulted in an overhang 50 ft wide on the north side. At this time the open stope was filled by surface mining and has been kept approximately full ever since. When surface mining dropped to an elevation below 3650, it was not possible to keep the hole full below this overhang, whereupon the overhang caved and the surface hole assumed the shape of the upward projected limits of the original block.

When the stope completely caved, in reality it became an oversized ore pass 60x100 ft in area and up to 241 ft in height.

Costs to date have checked with the estimate that 3650 elevation is the break-even spot between overland and underground haulage.

With this ore pass, so-called, in successful operation it has been possible to extend pit plans to include much of the ore reserve lying below the limits of economical trucking operations. Several locations

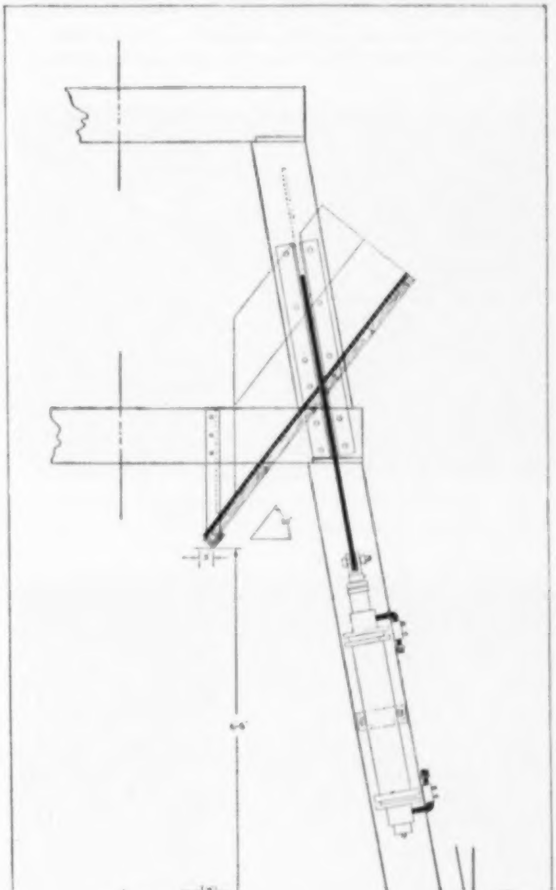

Fig. 2—Assembly drawing of guillotine-type chute gate installation.

Fig. 3—Dozers and carryalls working in cut below 3650. Transfer (dotted outline) full and not being drawn. Loads dumped on top of transfer. Shovel in background stripping waste on 3700 bench.

Fig. 4—Transfer being drawn. Loads dumped at edge to be pushed in by dozers. Cut below 3650 is shown.

Fig. 5—A load approaching side of the transfer for dumping while the transfer is active.

Fig. 6—Trucks dumping into transfer from 3700 bench. The 3650 bench is shown in background and the cut made by dozers and carryalls below 3650 at left center.

Fig. 7—A truck is shown dumping into the transfer from the 3700 bench. The 3700 bench and the scraper cut below 3650 appear at the right.

for similar blocks will be available as the Thornton pit progresses. As the 600 level is passed in the downward mining of Thornton pit, the 800 level will become the haulage level. Similar caved block ore passes are planned for the Live Oak area, where the 600 haulage level will serve for the life of the pit.

Mining Methods Used With Caved Block Transfer

Glory hole mining is an old method. Ore transfers, of limited cross section, to transfer surface-mined ore to underground are not new, the method having been used by United Verde in Jerome, by Phelps Dodge at the Sacramento pit in Bisbee, and probably by others. Through necessity Bagdad engineers used a caved block as an ore pass when developing their open pit. But so far as is known, no other company has deliberately caved a block into an open pit for the purpose of using it as an ore pass.

Since the opening is an ore pass it is desirable to maintain the walls as nearly vertically above the stope as possible. The mining method does not correspond to glory holing, as the hole is not progressively enlarged and emptied until the angle of repose is reached. Beyond a distance of approximately 1000 ft standard open-pit mining will be used, with trucks dumping into the transfer. This has been tried and found satisfactory.

Experience indicates that ore within 1000 ft of the transfer can be handled by bulldozers and carryalls, Fig. 3, at lower cost than by shovels and trucks. With this method, regular 50-ft churn drill holes are drilled and blasted in a 20-ft center pattern. Multiple lines are blasted and a channel is cut, with caterpillars and carryalls, sloping toward the transfer. The sides of this cut are kept on a 45° angle. When the cut has reached a depth of 50 ft, or less if desired, churn drill holes along the sides are blasted toward the cut in single or multiple rows and the ore cleared out again to a 45° angle. This process is repeated until the economical limits of the caterpillar and carryall are reached. Shovels and trucks take over at this point.

Drawing in a transfer of this area produces almost immediate subsidence on surface. There is little or no chance of a hang-up with its subsequent abrupt subsidence. Actually, the carryalls can run out over the full transfer and dump their loads while the ore is being drawn from below. For safety's sake, however, no equipment is allowed to cross the transfer while it is being drawn. Present practice is to draw the ore on one shift, Fig. 4, and refill the transfer on another shift, Fig. 5-7. When it is necessary to work on surface on the same shift in which ore is being drawn below, the carryalls dump their loads at the edge of the transfer and the ore is then pushed in with a bulldozer.

Four D-8's and two Model 90 caterpillar carryalls are the normal working equipment on surface. Five thousand tons per 8-hr shift can normally be transported by this equipment. Spare equipment consists of two smaller carryalls. Three other D-8's, normally assigned to other work, back up the motor power as spare units. In addition to lowering costs, use of this type of equipment for all ore within 1000 ft has released shovel and trucks to necessary stripping.

All truck haulage to the transfer will be a level haul. This should cut down on truck maintanence, and it is obvious that fewer trucks will be needed to transport any given tonnage. As the benches are lowered the transfer will be cut down 50 ft at a time, and it is quite probable that one side of the transfer may be 50 ft higher than the other. This will allow ore haulage from two benches.

In a test run with the shovel located 400 ft from the block, 6498 tons were truck-hauled into the block in a single shift. Only two trucks were used. Conditions were ideal and continuous production at this rate could not be expected.

It has been found that the ore breaks very well and little sizing is necessary on the grizzly level. Undoubtedly there is some crushing action in passing downward 240 ft. The normal crew for drawing and hauling consists of 3 men in the stope with from 8 to 10 men on the haulage crews, depending on the number of trains used. A 3-train operation is probably the most efficient. With 24 4½-ton car trains and 3 trains in operation on the 2500-ft haul to the shaft, as much as 6150 tons have been drawn in an 8-hr shift. A 5200-ton average per shift under these conditions has been maintained over a period of 2 weeks.

For the six months ending July 1, 1953, an average of 124 tons of ore per manshift was obtained in the combined operation. In the open pit this includes all shifts spent on drilling and blasting and transportation to the block, part by cabs and carryalls and part by shovel loading and trucks. Underground shifts include chute tapping, hauling and hoisting, and such repairs as were needed. Also included are miscellaneous shifts prorated to various mining operations on a shift basis, such as supervising, engineering, timekeeping, and ventilation. Two hoistmen are included. The hoistmen and most of the miscellaneous shifts, although their services are charged to the operations, would be necessary even if no ore were hoisted.

The draw from the transfer is not always regular, as this particular ore is used as a balance in obtaining both the desired grade and oxide-sulphide ratio of plant feed.

Repairs on the grizzly level and in the drawing area have been light. One timberman part time has been able to keep up with the necessary work.

It has been found that the steel grizzly sets are not much better than timber sets, but steel-control sets are almost essential. The air-operated doors on the haulage level are especially satisfactory. With 3 pairs of chutes it is seldom that ore is not available at each chute position so that 3 cars can be loaded simultaneously. With the development of subsequent transfer blocks undoubtedly improvements will be incorporated, based on experience gained with this initial block. Such changes will be mainly in re-enforcing points of wear and streamlining the flow of ore. This first transfer block up to July 1, 1953, has passed about 1¾ million tons of ore.

Summary

The use of a caved block for a transfer raise at Inspiration has 1—solved a sizing and transportation problem, allowing a sizeable increase in tonnage that will be available for mining by open-pit methods; 2—decreased costs on all ore originally planned for open pit lying below the 3650 elevation; 3—solved the sizing problem without installation of a crusher; 4—made use of existing facilities and equipment which were idle, and 5—released shovels and trucks for other work.

Acknowledgment

The writer wishes to thank members of the operating staff at Inspiration for their help in preparing this paper.

Mining and Milling of Lithium Pegmatites At Kings Mountain, N. C.

by E. R. Goter, W. R. Hudspeth, and D. L. Rainey

THE area in which spodumene-bearing pegmatites occur extends from Gaffney, S. C., in a northerly direction to Lincolnton, N. C., a distance of about 16 miles. The zone averages 2 miles in width.

Interest in this area was first aroused by the discovery of small amounts of cassiterite in the pegmatites and in greissen which occurs on the walls of some of the pegmatites. Attempts were made from 1880 into the 1920's to mine and concentrate cassiterite, but there were no successful operations. In 1935 L. M. Williams became interested in the area and began prospecting by trenching and sinking several small shafts. He gradually gained control of a considerable acreage south of the city of Kings Mountain, N. C. Mr. Williams states that he produced some ore from a shaft which he shipped to the Maywood Chemical Co. in the late 1930's. In 1937 G. H. Chambers of the Foote Mineral Co. visited the area and examined several properties, but the lithium market was too small to support the necessary concentrating plant.

Early in World War II the Solvay Process Co., encouraged by various federal agencies, acquired the Williams properties and leases and started a drilling program to develop reserves for spodumene mining. A study of this area was also made under T. L. Kesler by the U. S. Department of the Interior Geological Survey.

Starting in 1942 a concentrating plant, water supply dams, and shops and other buildings were constructed. The Superior Stone Co., a neighboring crushed stone operator, was awarded a contract to mine and crush the pegmatite ore. Late in 1943 the plant started operations, which were continued until February 1946. A drop in demand for lithium chemicals forced it to close and remain closed for several years.

In 1948 the Foote Mineral Co. became interested in assuring itself a source of lithium ore on the North American continent, to be used in the production of lithium ore chemicals, and inspected properties in Canada, South Dakota, and North Carolina. The North Carolina deposits were the most promising. In the fall of 1950 the company leased the Solvay interests with option to buy and acquired additional properties. Renovation of the mill and preparation of the quarry for mining was begun in earnest in January 1951.

A new crushing plant was built and construction started in July 1951. The flotation process used was developed in the North Carolina State Minerals Research Laboratory located at Asheville, N. C.

Foote is now in the process of expanding its production. The crushing plant is being revamped and much of the old milling equipment purchased from Solvay will be replaced by new. Mining equipment of suitable capacity has been received and is in use. The frontispiece is a general view of the mill.

The orebodies consist of pegmatite dikes composed of fine-grained albite and quartz, with varying amounts of mica and spodumene. The latter may amount to as much as 50 pct of the rock, but the overall average is about 15 to 20 pct. Tin, columbite-tantalite, beryl, and other valuable minerals are present in varying small amounts.

These dikes, which intruded into the Roan Gneiss underlying most of the Piedmont area, strike approximately N 30 E and dip about 85° to the west. The gneiss has the same strike but dips about 70° west. The dikes lie in a belt extending from Lincolnton, N. C., to Gaffney, S. C., and passing through

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the town of Kings Mountain. For years this was known as the Carolina tin belt and was worked in numerous places for cassiterite. The tin content was spotty, however, and no large-scale production was ever achieved.

The first published account of tin in the belt appeared in 1884,[1] but from that time until as late as 1931 the existence of spodumene is scarcely mentioned except as a constituent of the tin-bearing pegmatites. By 1942 one small quarry was opened for the mining of this lithium ore.[2] According to Mr. Kesler this pit was 100 ft long, 12 ft wide, and 35 ft deep.

During the period from 1942 to 1944 the Solvay Process Co. which operated this property dug eight trenches for a total of 1600 ft to determine the extent and depth of the orebodies. One trench was about 650 ft long and crossed three sizable dikes. Two churn drill test holes were put down for a total depth of 270 ft. Seventeen inclined diamond drill holes were drilled, a total footage of 10,011 ft. The deepest of these was stopped when it had reached a vertical depth of about 750 ft.

Since beginning operation in 1951 Foote Mineral Co. has carried on a continual program of prospecting. Considerable detailed geologic mapping has been done, necessitating 2000 ft of trenching by bulldozer. A program of diamond drilling is being conducted to supplement information obtained from core holes drilled by Solvay Process Co. and to outline more accurately the orebodies at depth.

Past and Present Mining

The quarry in existence in 1942 was on the east side of pegmatite No. 25 which was 85 ft wide at the top and believed to be at least 1450 ft long. This became the starting point for Solvay Process Co. operations. They increased the pit to about 400 ft long, a maximum of 60 ft deep, and the full 85-ft width of the dike. Mining at this time consisted of drilling the rock in 20-ft benches with wagon drills. After the large rocks were blasted and blockholed, the ore was loaded by power shovel into trucks and hauled to the crushing plant. Benches were carried across the width of the dike and the face was advanced along the strike.

Solvay also began mining the north end of pegmatite No. 21 at a starting point 60 ft wide, advancing the quarry as a side-hill operation by going in from one end and carrying the bottom level. At an advance of 300 ft the quarry had obtained a maximum depth of 40 ft and was being worked from two benches. This pit was abandoned because the ore was weathered and an acceptable grade of concentrate could not be made at that time.

Foote Mineral Co. began operations by dewatering the quarry in pegmatite No. 25 and continued with much the same procedure Solvay had used. By October 1952 the pit had obtained a depth of 120 ft and was over 700 ft long. Including stripping to obtain safe walls, it measured about 150 ft wide at the top. The ore had narrowed to less than 50 ft wide at the bottom. Since the ore pinched in very rapidly in the south end, and a *horse* of waste in the north end made advancement in that direction uneconomical, working length was limited. The quarry in the north end of pegmatite No. 21 was also cleaned out, but it was found that this ore still could not be successfully milled.

To open a new quarry, the top, end, and sides of pegmatites No. 18 and 21 were stripped at the south end for a distance of about 300 ft. Stripping bore out indications that these two dikes were very near together. To obtain a longer working face and therefore a more flexible operation, it was decided to mine across the strike of the orebody. Accordingly a face approximately 200 ft long was opened along the footwall side where the country-rock had been stripped back for a distance of 50 ft to a depth of 20 to 30 ft. However, because of the weathered condition of the ore and the presence of much clay in the seams, a large portion of this ore had to be discarded as waste. Even then the mill had difficulty making a good product.

An additional quarry was opened on the largest of the pegmatite bodies in the area lying some 2500 ft north of pegmatite No. 25. This dike, known as pegmatite No. 8, has a maximum width of 300 ft and is well over 1000 ft long. Because of the topography it was possible to develop a face 200 ft long by clearing the trees and making a road along the footwall. At the present time mining is being carried on simultaneously with stripping operations. The top 6 to 8 ft is being wasted, as it contains most of the dirt and clay so detrimental to the milling operations. The shovel operator loading ore for the crusher, Fig. 1, sorts it to remove as much of the waste as possible. This material is then dumped over a scalping grizzly to remove any remaining dirt. The resulting ore, while stained, is free of the clay and can be successfully concentrated. The first bench in the pegmatite No. 8 quarry started out at 16 ft, but because of the rising ground it will reach a height of about 30 ft. Subsequent benches will be kept at 20 ft. This is due to the fact that the fractured nature of the orebody makes the use of higher benches unsafe.

Present equipment in the mine includes air compressors with a combined capacity of about 900 cfm, two 4-in. wagon drills for drilling benches, two jackhammers, and one 3-in. wagon drill used in stripping. A Northwest 25 shovel and a Bucyrus-Erie 38-B shovel are used for both stripping and a loading ore, and a P & H 255A crane with a 40-ft boom and 4000-lb ball is used for secondary breaking. Two 10-ton Euclid trucks are used for hauling, supplemented by two 2-ton GMC trucks hauling on a contract basis. A TD 18 with hydraulic dozer blade is used for clearing, cleaning, and roadwork.

Future development will, of course, depend upon the character of the dikes being worked. There are no plans at present for any underground operations, since there is at least 10 years' reserve of ore for surface mining. Orebodies in future will be developed by stripping the eastern footwall to give a 30-ft

Fig. 1—Loading rock in open pit for trip to jaw crusher.

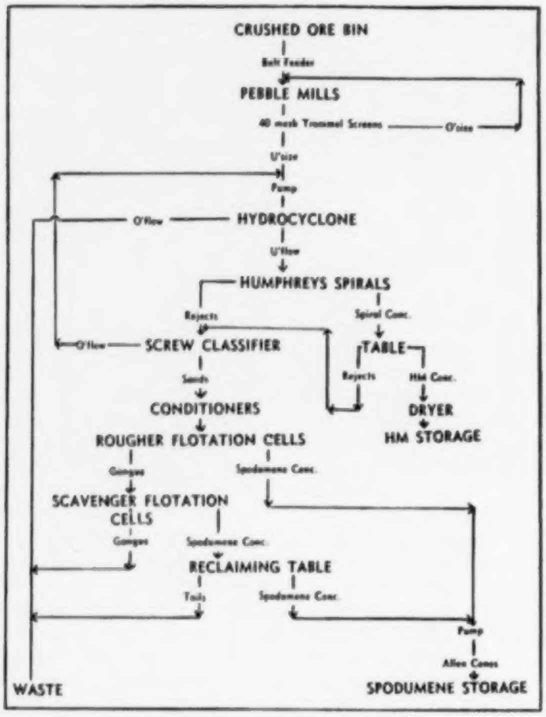

Fig. 2—Mill flowsheet.

roadway a slope of about 1 to 1 in the waste. The face will then be carried across the strike of the dike and the hanging wall sloped at whatever angle is necessary for safety. This will give a safer operation than the method used in pegmatite No. 25 and also make it possible to recover some ore from small dikes which lie close to the larger orebodies.

Ore from the mine is dumped onto a 10-ft by 36-in. Telsmith apron feeder, which feeds an 18x36-in. Traylor jaw crusher. The −3 in. jaw crusher product is delivered by conveyor to a 4x10-ft Ty-Rock screen clothed with ¾-in. Ty-Rod screen cloth. Screen undersize is delivered by chute to a bucket elevator which discharges into the crushed ore bin. Oversize from the screen is fed to a 436 Allis-Chalmers Hydrocone crusher, set to discharge a product about 66 pct −¾ in., which is returned to the primary conveyor feeding the Ty-Rock screen. A mill flowsheet is shown in Fig. 2.

A belt feeder equipped with a variable speed drive delivers the crushed ore at a rate of about 360 tons per day to two 6x8 Hardinge pebble mills, lined with 30-lb rail and charged with 3x4-in. granite blocks as grinding media. The mills are driven with 125-hp induction motors, and grinding charge is maintained by adding media until the motors are pulling 150 amp at peak of surge. The mills are equipped with trommels clothed with nominal 40-mesh Ton-Cap screen. About 20 pct of head feed reports as screen oversize in the mill discharge and is returned to the heads of the mills by an Esperanza drag. Screen undersize is laundered to the pump-box of a 4-in. Wilfley pump.

The screen undersize is pumped at about 15 pct solids to a 12-in. hydrocyclone for desliming. The hydrocyclone is centered over a round gathering box with eight wedge-shaped compartments which empty into the feed boxes of eight 5-turn, 24-A Humphreys spirals. The cone-shaped spray discharge of the hydrocyclone effectively distributes underflow to the spirals. About 5 pct by weight of the spiral feed is removed from the ports as a rougher heavy mineral concentrate and delivered to a table for further concentration. The spirals rejects are laundered to a 21-ft x 48-in. Wemco spiral classifier, which discharges the sands into No. 1 conditioner.

Flotation

Two conditioners in series are operated at about 55 pct solids. The following reagents are added at this point: amine, a silicate gangue mineral collector; caustic, a pH regulator; dextrine, a selective depressant for spodumene in an alkaline circuit; and pine oil as a frother. It has been found that a delicate balance exists between the amine and the pH in the conditioners. Dextrine depresses most effectively in a highly alkaline circuit; the collecting properties of the amine, however, are sharply reduced when the pH of the pulp goes above 11.3. The conditioner discharge is diluted to about 25 pct solids and delivered to the feed box of a bank of six No. 24 Denver flotation cells. In this separation the float product contains the gangue minerals: quartz, mica, feldspar, and some hornblende. The underflow is the spodumene concentrate which is pumped to an Allen cone, dewatered, and delivered to storage. The rougher gangue froth containing about 7 pct spodumene is delivered to a bank of six No. 22 Morse-Weinig cells for scavenging. The scavenger machine discharge, usually not of concentrate grade, is pumped to a Deister table for gravity concentration. The table tails are pumped to waste and the table concentrate is combined with the rougher cell spodumene concentrates.

For mill control purposes, the spodumene content of any given sample is determined as follows. A weighed sample is placed in a centrifuge tube containing acetylene tetrabromide, sp gr 2.96. The tube is spun and the spodumene, with a specific gravity of about 3.2, sinks in the liquid. The floats are removed and the sinks are filtered, washed with benzene, dried, and weighed. The percent sinks is roughly equivalent to the percent spodumene. An error is introduced, however, if any quantity of hornblende is present.

The concentrates drain to about 95 pct solids on the storage pads, see Fig. 3. A scoopmobile transports the concentrates to a hopper from which the material is fed to a Ruggles-Coles rotary drier. Spodumene concentrates to be used in the manufacture of ceramics are passed over an induced-roll

Fig. 3—Spodumene on draining pads awaiting drying and bagging operation.

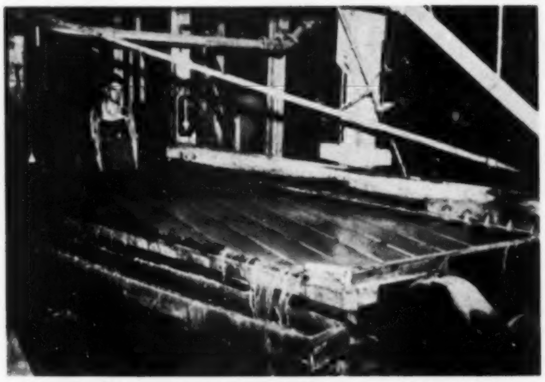

Fig. 4—Shaker table for recovery of tin columbite by-products of spodumene process.

type magnetic separator, where iron and iron-bearing minerals are removed. When the iron content is not critical the concentrates are loaded and shipped immediately after drying.

At present a heavy mineral concentrate is the only by-product being made. The rougher spiral concentrate mentioned before is delivered to a Deister table, Fig. 4. The table concentrate consists of cassiterite, columbite, and tantalite, with small quantities of pyrrotite, pyrite, and monazite, and is produced at the rate of approximately 80 lb per day. Table rejects are pumped into the spiral classifier and returned to the flotation circuit.

Feldspar concentrates have been made in the plant by use of amine, sodium fluoride, and sulphuric acid, but the iron content in the concentrates has been over a commercial limit because of the presence of hornblende. The iron-bearing minerals can be removed by magnetic separation. Another deterrent to plant-scale production of feldspar is that with the relatively fine grind necessary for the beneficiation of spodumene the feldspar is very fine and consumer prejudice against a product of this nature will have to be overcome.

The mica content in the head feed varies from almost nothing to about 2 pct. For a time a mica concentrate was produced by removal of the slimes portion from the spirals used for heavy mineral production. Several tons of a commercially desirable mica concentrate were produced per month by this method. However, the operation of a spiral for mica production and for heavy mineral concentration is not the same. It was found that heavy mineral production was impaired and the production of mica was temporarily abandoned.

The Foote Mineral Co. is cooperating with the Bureau of Mines in an intensive investigation of recovery of beryl from the pegmatites. This work shows some promise and will be tried on a plant scale as soon as conditions permit. The beryl in the pegmatites is of the clear variety and is very difficult to identify.

Construction has begun on an expansion program which will raise the capacity of the operation from the present 360 tons per day to 1000 to 1200 tons per day. Changes to be made in the crushing plant include the installation of a 26-in. Traylor primary gyratory crusher and the conversion of the present 436 Allis-Chalmers Hydrocone crusher to a 636 crusher for the production of 1¼-in. material.

Two 5x12-ft Marcy rod mills will receive ore delivered by two Hardinge constant weight feeders. The discharge of the rod mills will be delivered to 54-in. Wemco spiral classifiers, which will make a separation at approximately 40 mesh, the screw sands returning to the mills. Classifier overflow will be deslimed by hydrocyclones and the sands passed over 32 Humphrey spiral classifiers for recovery of tin and columbite. A Wemco classifier identical with the one presently in use will receive the spiral rejects from one set of spirals. Two banks of No. 24 Denver Sub-A Cells will receive the conditioner discharge and present equipment will be placed in service for scavenging the rougher gangue froth products.

The very desirable feature of the expanded milling facilities will be the two parallel circuits, each capable of handling 500 to 600 tons per day. This will allow proper scheduling of maintenance and partial production in event of equipment failure.

Power, which is furnished by the Duke Power Co. at 44,000 v, is reduced to 440 v for distribution to the plant. Where the power lines to the water-pumping stations and compressors are long, the voltage is again stepped up to 2300 v. Newer equipment is being installed with 2300-v motors.

Tailings disposal is conventional, with an earth tailings dam and Weir-type overflow into culverts buried in the dam. The overflow Weirs are 24-in. diam concrete pipe, the first one having been set in a concrete housing which conducts the overflow into the discharge culverts. The Weir is raised by addition of 1-ft sections of concrete pipe as needed.

Service Departments

Repair, maintenance, and construction are organized as a separate department, headed by a plant engineer and directly supervised by a maintenance supervisor and two foremen. Except for the maintenance supervisor, who had been an electrician for Solvay Process Co., the majority of the personnel in this department had no experience in the mineral production field and were trained after being hired. It has been necessary in the past to contract major construction jobs, such as the building of the crushing plant. It is planned that in the future much of this can be taken care of by the maintenance department. The plant was idle for 6 years and had been furnished largely with second-hand equipment. The 24-hr 7-day operation in the mill placed a heavy burden on this equipment, with resulting high maintenance charges.

An employee welfare supervisor heads the safety committee, which holds monthly meetings and makes regular inspections. Accidents that have occurred are discussed and future prevention outlined. Protective clothing is furnished when needed.

All shipping is done from a loading station located on the main line of the Southern Railroad about 1½ miles from the plant. Here a hopper-conveyor belt-bucket elevator system is used to load bulk shipments into hopper cars. It is also used together with a car loader to load box cars. A loading platform is utilized in loading other shipments. Both bulk and bagged products are hauled to the loading station by truck.

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Economics of Pegmatites

by Paul M. Tyler

MUCH information concerning pegmatites which was thought to be true a few years ago has been proved false, and what is now actually known about some pegmatites is not true of many others. The erratic and seemingly unpredictable structure and variable composition of this class of mineral deposits has been widely emphasized. Even parts of the same pegmatite body exhibit marked differences in texture, mineral composition, width, and attitude. Constructive geological thinking in respect to pegmatites now aims to establish general laws rather than to stress the confusing diversity of features having no special economic significance.

Substantial progress has been made in classifying different types of these deposits according to general features, internal structure, mineralogy, and origin. In some cases it has even been possible to block out tonnage reserves in advance of mining. It is still easy, however, to make highly erroneous predictions after a preliminary examination of a pegmatite prospect.

Pegmatites are important to the economic well being of the country and to its military security. They furnish virtually all the feldspar, strategic mica, beryl, columbium, tantalum, and caesium utilized in the United States, as well as sundry other minerals and significant amounts of lithium and rare earth minerals and gems. With the exception of vermiculite, occasional ilmenite-rutile, and perhaps soda-lime feldspar and garnet deposits, basic pegmatites are of little economic importance. Consequently in this paper, as in common parlance, the term *pegmatite* generally relates to coarse-grained acidic rocks or what is aptly called *giant granite*.

Available data indicating the size and importance of the production and trade in specified pegmatite minerals are summarized in Table I.

Geological Features

Much of the latest thinking on the economic geology of pegmatites is now available in a 115-page monograph[1] by a group of experts who participated with geologists of the Federal Geological Survey in the widespread wartime investigations. Doubtless the most significant feature of the monograph is indicated by the title, *The Internal Structure of Pegmatites*, but it also contains a vast amount of other new information and includes the assimilated concepts of many earlier writers, whose works are given in a comprehensive list of references.

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The shape, size, attitude, and continuity of many pegmatite bodies is controlled by the structure of the older rocks in which they occur. If the older rocks are easily penetrated, e.g., biotite schist, most of the pegmatites in a given district will be found outside the parent granite mass as exterior pegmatites. Marginal pegmatites are more prevalent if the older rocks are massive, unsheared, and sparingly jointed. Networks of pegmatites are abundant in highly-jointed rocks. In strongly foliated schists the bodies are usually lenticular, whereas in highly-folded areas they assume tear drop, pipe or pod-like, bow-shaped, or sinuous forms. Jahns[2] recognizes five major shape classes: 1—dikes, sills, pipes, and elongate pods; 2— dikes, sills, pipes, and pods with bends, protuberances, or other irregularities; 3—trough-or scoop-shaped bodies with or without complicating branches; 4—bodies with the form of an inverted trough or scoop; and 5—other bodies, including combinations of the above and miscellaneous shapes.

Many pegmatite deposits are scarcely big enough to be recognizable as such. Most of them, in fact, are small tabular deposits less than 4 in. wide and usually without economic concentrations of minerals. On the other hand, some pegmatites are more than a mile long and over 500 ft wide. The ratios of length to breadth range from 1 : 1 to 1 : 100. Although the vertical dimension bears no invariable relationship to strike length, tabular deposits or large lenses are often symmetrical enough to show nearly as much continuity down dip as in their horizontal extension, and some pipes or pods are amazingly persistent in the vertical plane.

Small pegmatites often string along a fairly definite trend line; in a given district major bodies may lie roughly parallel, and where only a few of them do not, the erratically disposed bodies generally differ in composition from those conforming to the regular pattern. This does not apply, however, in all districts.

Characteristically, pegmatite veins pinch and swell or split into branches. When they pinch out entirely it is often possible to find a new body by prospecting the extension of the strike or dip, but the chances of finding a hidden deposit are ordinarily too uncertain to justify much subsurface prospecting. Diamond drilling may yield valuable information as to the continuity of known deposits whose upper portions are well-exposed. Some deposits, in fact, can be proved up for hundreds of feet by surface trenching and then intersected by drill holes at various depths like any other vein-like deposit. Others twist and branch, apparently defying all efforts to explore them short of actual mining.

The internal lithologic and structural units that can be recognized in many pegmatite bodies have been listed by Jahns[2] as follows: 1—fracture fillings, generally tabular bodies that fill fractures in previously consolidated pegmatites; 2—replacement bodies, formed at the expense of pre-existing pegmatites with or without obvious structural control; and 3—zones or more or less complete shells whose boundaries may be roughly parallel to the outer walls of the main body. These three types of units are distinguished by differences in mineralogy or texture or both. The contacts between units may be knife-edge sharp or diffused over several feet of broad gradations, and they need not be uniform.

Some pegmatites seemingly lack systematic orientation, and detailed mapping and careful observation are required for the observer to recognize an orderly arrangement of structural units. Zoning may be incomplete. It may be absent in parts of the body and develop on only one side or at one end. Instead of concentric shells, zoning may exist only as lenses, layers, or elongated pods.

Perfect zoning would comprise a series of concentric shells surrounding a central zone or core. Heinrich,[3] for example, concluded that zones may be formed by crystallization of the pegmatite magma in successive layers inward from the contacts. The coarser texture of inner zones can thus be explained by slower cooling.

The four principal types of zones have descriptive names: border zones, wall zones, intermediate zones, and innermost zones or cores. Any of these, especially intermediate zones, may be absent or virtually undeveloped. On the other hand, some pegmatites contain several intermediate zones.

Fifty or more minerals have been reported as occurring in pegmatites, and specimens of as many as 30 varieties are frequently found in the same body. Studies of the relation between structural features and sequence of mineralogical units have indicated that the mineral assemblages are related to zoning and change in approximately the following order, starting from the wall rock and progressing inward: 1—plagioclase, quartz, and muscovite; 2—plagioclase and quartz; 3—quartz, perthite, and plagioclase with or without muscovite and/or biotite; 4—perthite and quartz; 5—perthite, quartz, plagioclase, amblygonite, and spodumene; 6—plagioclase, quartz, and spodumene; 7—quartz and spodumene; 8—lepiodolite, plagioclase, and quartz; 9—quartz and microcline; 10—microcline, plagioclase, lithia mica, and quartz; and 11—quartz.

Few, if any, pegmatites contain all 11 of these zones, but those that are present are likely to be grouped in this order. As indicated by this arrangement, the processes of albitization, muscovitization, and the development of lithium minerals are important stages in the formation of complex pegmatites. Other accessory minerals also show a similar sequence. Beryl, for example, may be more abundant in the wall zone, lean in the intermediate zones, but occasionally present as a single giant crystal or nests of crystals in or near the core. Undoubtedly the small crystals and more or less corroded anhedral grains of beryl associated with plagioclase, mica, and quartz in and near the wall rock were formed first, whereas second-generation beryl is represented by the more familiar occurrence of large anhedral or corroded crystals associated with albite, potash spars, quartz, muscovite, and lithium and/or tantalum minerals in zones parallel to the outer edges of very coarse-grained cores.

In two-zone pegmatites, the cores may comprise medium-to coarse-grained plagioclase, potash spars, quartz, mica, and tourmaline. Some of the most productive mica deposits are composed principally of coarse plagioclase and quartz. In general, however, the present writer believes that pegmatites with relatively simple zonal structure or none at all are poor in commercial minerals other than feldspar and that unless albitization is present the search for other minerals including strategic mica books may be unrewarding.

Useful Minerals in Pegmatites

Feldspar: The bulk of tonnage output of all varieties of pegmatite minerals is feldspar. This is the main constituent of most pegmatites and the feldspar-rich zones are commonly the largest and most persistent. Granitic pegmatites are still the principal sources of commercial production, although other occurrences, including massive aplites and coarse-grained granitoid rocks, are coming into the picture. About 60 pct of the composition of all igneous rocks is made up of feldspars, but petrographic examination or chemical analysis shows that feldspar grains in most unpegmatized rock masses are heavily shot with iron-bearing minerals. Another limitation on the number of deposits that can be utilized as commercial sources is the fact that the spar in many igneous rocks is anorthite, or some other lime-rich refractory variety, rather than albite or the even more generally useful potash spars.

Commercial feldspar deposits commonly form cores in potash-rich bodies. In some mines, however, the potash-rich core is more or less surrounded by plagioclase-rich zones, one or more of which may be a productive mica deposit. Although high-grade deposits of both mica and feldspar are

Table I. United States Production and Imports of Pegmatite Minerals

	Production, Mine Shipments						Imports for Consumption	
	1940 to 1944, Avg		1945 to 1949, Avg		1951		1951	
	Short Tons	Value, $	Short Tons	Value, $	Short Tons	Value, $	Short Tons	Value, $ *
Feldspar	354,228	1,559,673	487,401	2,373,879	448,492	2,815,587	19,183	146,565
Mica**	22,014	1,978,157	35,701	963,169	69,650	1,868,544	7,639	3,945,711
Lithium minerals	6,544	241,510	3,334	259,457	12,897	896,000	†	†
Beryl	258	27,150	146	37,361	439	14,500	4,316	1,366,772
Columbium-tantalum ores	3	10,743	2	6,225	‡	1,528	888	1,552,776
Tin	28	28,160	17	32,958			64,822	157,019,209
Total	383,075	3,845,393	526,601	3,673,049	531,478	5,596,159	96,848	164,030,933

* Foreign market value does not include freight to United States, import duty (if any), or other costs of importation.
** Includes scrap mica and mica recovered from kaolin and schists.
† Not reported separately.
‡ Less than 1 ton.
Source of information: U.S. Bureau of Mines.

likely to occur in well-zoned pegmatites, the portions of a pegmatite that contain the most mica are generally less rich in feldspar, and vice versa.

Whereas feldspar is a potential joint product of almost all pegmatite operations, the feldspar industry has not hitherto been a major source of other minerals. A good deal of feldspar is still produced by selective mining, the muck being handled with a fork to reject fines. A substantially growing proportion of the total domestic output, however, is coming from flotation plants processing run-of-the-mine pegmatite or alaskite, and some of these plants produce flake mica and perhaps quartz. Since most of the best feldspar deposits are not rich in other economic minerals, and since other economic minerals, if present at all, generally occur in parts of the deposit which may not pay to work for spar, the feldspar industry is a less important potential source of byproduct minerals than is generally supposed.

Despite occasional shortages caused by swiftly rising demands, mainly in the glass industry, supplies of feldspar in the United States are so enormous that production can respond quickly to any foreseeable increase in demand. In the event that byproduct production of feldspar caused a sharp reduction in prices, demand would probably increase, but further expansion in normal requirements is more likely to pace the general level of industrial production.

Mica: Strategic mica comprises sheet or block mica in the rifted condition (roughly split and trimmed), No. 6 (about 1 by 2 in.) or larger in size, of *stained* or better quality, having suitable electrical properties, and relatively free from cracks, pinholes, reeves, waves, and other mechanical defects. No suitable substitute has yet been found for the fine-grade mica used in high-duty capacitors and vacuum tubes, essential components of automatic measuring instruments, control circuits, and electronic devices used for military and civilian communications. Only the highest quality of mica is acceptable for government stockpiling, and mica from many mines will not meet specifications for use in capacitors set forth in ASTM Designations D-351 and D-748.

Muscovite is of widespread occurrence in common granites and schists, but only in small flakes. Schists composed principally of mica are significant sources of ground mica, and more and more mica is reclaimed as a byproduct of milling kaolin, kyanite, and several other nonmetallic minerals. Strategic mica, however, is found only in pegmatites, and even in the best mines, this is an erratic and minor constituent. The total yield ordinarily ranges from 2 to 7 pct of the rock broken. Moreover, only about 3 to 8 pct of the mine-run mica produced at domestic mines during World War II was finally obtained in the form of full-trimmed block. The yield of strategic mica is frequently less than 1 lb per ton of rock broken.

Following the uncertain and laborious operations involved in finding workable deposits, exploring them for mica-bearing shoots and extracting the mica-bearing rock, there is still a vast amount of work to be done to prepare it for market. Most of the hand-cobbed mica, at least 90 pct and often 95 pct or more, is necessarily rejected in the subsequent rifting, trimming, and splitting operations. It goes to the fabricator in the form of irregular pieces which must be reclassified and re-sorted before going to punching machines. Stampings can be made only one at a time, the operator holding the small piece of mica in her fingers. Even with careful selection of raw material and efficient operators at the punching machines, at least 70 pct of the weight of the closely trimmed mica sheets is wasted in manufacturing. In short, as little as 0.3 lb of mica stampings is normally obtained from a ton of rock actually mined.

From prospecting to final fabrication and inspection of products, the mica industry thus fails to fit into the modern pattern of mass production. Despite these impediments to the establishment of the industry on a firm foundation, the United States must have mica, and in large amounts. There are now insistent demands for defense production and stockpiling. Prices offered for strategic mica from domestic mines are many times higher than ever before and programs for augmenting supplies from India, Brazil, and certain other important foreign sources are being pushed forward energetically.

Lithium minerals: Although lithium is recovered in significant quantities as a minor constituent of saline deposits, the pegmatite minerals, spodumene, amblygonite, and lepidolite, are most important in meeting the fast-growing requirements of this element and its compounds. In South Dakota and North Carolina there are measured reserves in many tens of thousands of tons and inferred reserves in millions of tons. The numerous large deposits near Kings Mountain, N. C., are remarkably persistent and the spodumene content amazingly uniform, almost invariably exceeding 20 pct. In contrast to conditions at other commercial pegmatites sources, however, the texture of these deposits is such that froth flotation rather than hand picking is necessary to obtain a reasonably clean product. Potential byproducts—in addition to the usual feldspar, mica, and quartz—include small amounts of beryl and traces of the heavy minerals, cassiterite and columbo-tantalite.

Notwithstanding the enormous resources in several states, domestic supplies of lithium minerals usually have been imported. Southwest Africa has been a major source in recent years and now Manitoba has begun to develop its large deposits. Although still rated as a rare metal, lithium is one of the very few useful elements that can be obtained readily from pegmatites in vastly greater quantities, both in this country and abroad.

Beryl: In contrast to the situation with respect to lithium, procurement of beryllium has developed a long series of problems with no prospect of relief. The price lever has jacked up world production more than tenfold since 1937, but this is not enough. Fortunately, the wartime scare abated when graphite was substituted in the atomic stockpile, but prices again climbed rapidly, even before the Korean conflict, with the urgent demand for beryllium alloys for a variety of equipment needed by the armed forces and by civilians. The government now offers to buy clean beryl at $400 a short ton, following inspection, with the expectation that it may not average more than 8 pct BeO. This compares with a prewar price of $30 or less per ton for material containing a minimum of 12 to 14 pct. Doubts are expressed as to whether future world requirements can be met if prices go to $75 or more per short ton unit.

The United States, by far the principal consumer, has always had to seek its major supply of beryl abroad. Domestic production probably never exceeded 100 tons in any year before 1940, and al-

though there was bonanza strike in the Harding mine in Taos County, N. M. in 1950, the all time record annual output in this country has barely exceeded 500 tons as against requirements variously estimated up to ten times this figure.

Much of the world's beryl still comes from the beryl-tantalite mines in the bulge of Brazil. Argentina's output has been small since the famous Las Tapias mine was exhausted in 1941. This is probably the only mine that ever produced beryl as the main product. Stimulated by the short-lived embargo upon exports from Brazil, production was expanded in India and Africa. Currently several African sources seem promising, notably a long belt containing numerous pegmatite dikes in central Nigeria.

In French Morocco the present writer has seen ragged Berbers handcobbing beryl from rock fragments no more than half an inch across and sorting out pieces of beryl scarcely larger than a grain of corn. Even in low-labor cost countries, however, there is a limit to hand picking. There are fairly large known reserves of beryl in the outer zones of numerous domestic pegmatite deposits, but these are very low grade and the crystals so small that they cannot be unlocked except by fine crushing or grinding. Beryl is readily separated by froth flotation from most of its associated minerals except spodumene. Apart from unlocking disseminated grains and obtaining surfaces fresh enough to be filmed by flotation reagents in a pulp which has undergone kaolinization, the main milling problem is the minute volume of concentrates. The mill feed in most cases will not run over 1 pct of recoverable beryllium mineral, so the problem is economic as well as technologic.

The writer has not investigated rumors that beryl occurs in disseminated deposits in unpegmatized or only partially pegmatized rock masses in at least two places in the United States and also in Africa. With these possible exceptions, future supplies of beryllium evidently will depend upon utilization of the exceedingly low grade occurrences of beryl in pegmatite bodies. Some years ago attention was drawn to a New Mexican occurrence of helvite, but further prospecting failed to disclose enough tonnage or grade to permit commercial production.

Columbium-tantalum minerals: There is reason to hope that dependence upon pegmatites for columbium (niobium) and/or tantalum may some day be at least partially relieved. During the late war, the Germans gave attention to deposits of koppite in limestone, one in Norway and a smaller one in Germany. Considerable work was done on the Norwegian deposit after the war, but none of these investigations has resulted in commercial production. Currently, however, attention is being focused on occurrences of minute amounts of columbium in Arkansas bauxite and other deposits associated with nepheline syenites. This element is far more widely dispersed than previously supposed and may some day be recovered from residues of the Bayer alumina process and perhaps from other byproduct sources.

Since tantalite and columbite form an isomorphous series, ores from most localities contain both elements in appreciable amounts. The two elements, however, have different uses; consequently the mixed ores are considered less desirable than those containing a large excess of one or the other element. Columbite, for example, preferably contains a minimum of 45 pct Cb_2O_5 and the Ta_2O_5 content should be low; some contracts still specify a 10 to 1 ratio. The question of how best to handle mixed ores has not been solved, but the basic problem is to get enough columbium for its rapidly expanding field of use in welding stainless steel and for certain high-temperature alloys. Few pegmatites contain even recognizable traces of these minerals, and microlite, the only other mineral that can be termed an ore of tantalum, is producible only sporadically and in relatively small quantities.

Since domestic production of these minerals has never amounted to much, the United States has depended largely on imports. Hitherto the world supply of columbite has come mostly from Nigeria, where it is a byproduct of tin placer mining, and tantalite has come mainly from Brazil, where it is mined along with beryl. The Belgian Congo has recently become a large supplier of mixed ores which occur in decomposed pegmatites mined for tin. Smaller amounts of mixed concentrates are produced irregularly in other countries but known world sources are inadequate to meet the potentially heavy demand.

Miscellaneous minerals: Quartz, the almost universally abundant constituent of pegmatites, is often saleable. Crystals suitable for radio or other piezoelectric uses are almost never found in pegmatite, but pegmatite quartz is often sought for making crushed grains for coated abrasives and bitstone or for placing sand for potteries and special refractories. Hand-cobbed quartz has been employed as a substitute for Danish flint grinding pebbles, and coarse tailings from feldspar mills often qualify as glass sand. Crude or roughly crushed quartz is used principally for producing metallurgical flux, for making ferrosilicon, for filling acid towers, and for grinding into silica flour. For most of its uses, however, prices are low, often under $3 and rarely as high as $10 a ton.

A variety of gems occur in granitic pegmatites. Among the most important are the beryl group, tourmaline, topaz, spodumene (kunzite and hiddenite), quartz, and feldspar crystals. Rose quartz, smoky quartz, clear quartz, and amazon stone may occur in massive form or in large crystals in the main body of the pegmatite. Although usually outside the main feldspar zone they are recoverable at some feldspar mines as a byproduct. Other gem materials, however, tend to occur only in vugs or fissures or embedded in clay-filled cavities. Owing to their mode of occurrence and more especially to their fragility, gem materials cannot be considered a potential byproduct of ordinary pegmatite mining. Theoretically, many of them might report with other heavy minerals in a mass mining and milling operation, but even if they accompanied the more abundant minerals they might be badly damaged by heavy blasting and by the crushing operations. In mines worked for valuable gems, blasting must be stopped as soon as the gem-bearing zone is reached and further excavation accomplished mainly by chipping, wedging, and prying away enclosing rock. Even heavy sledging may induce the formation of incipient cracks.

Tin, tungsten, zircon, metallic sulphides, several uranium minerals, monazite, and even *gold* and *graphite* are minor constituents of various pegmatites, but commercial production of these minerals from pegmatite deposits is insignificant. The minor

elements *caesium* and *rubidium* do come from pegmatites, and in Madagascar and perhaps elsewhere complex yttria-bearing minerals, *gadolinite, cyrtolite*, and *fergusonite*, are produced on a small commercial scale as well as for mineralogical specimens.

General Economic Considerations

All who are familiar with problems of mineral supply are aware of the amazing record of the American mining industry in its struggle to hold down costs despite tremendous handicaps. Improved technology and better management have gone far to offset soaring wage rates and the necessity of working leaner and leaner ores at greater and greater depths. As this paper has already emphasized, increased understanding of the structure of pegmatites and the occurrence of these minerals has lessened the risks involved in prospecting and mining. Refinements in froth flotation, magnetic separation, electrostatic processes, and other methods of mineral dressing now permit much sharper separations of the minerals in mixed ores than seemed possible even a very few years ago. Last of all, there is government assistance in financing exploration under the Defense Minerals Exploration Administration. Other agencies advance loans against production and guarantee incentive prices for products.

Nothing is apparently wrong with this economic setting for a relatively small number of favorably situated major pegmatite deposits. For many years the present writer and others have advocated establishment of an enterprise based upon mass mining of pegmatite material, wall-to-wall mining in some cases, followed by suitable milling operations designed to recover all useful minerals from the ore. To test this objective government-financed demonstration plants are now seriously being considered in two or more districts.

Pegmatite operations that can be fitted into the broad pattern of American production should be able to prosper. The real question is how many pegmatite deposits or groups of neighboring deposits can be operated on a large enough scale or can furnish sufficiently uniform flow of mill feed to conform to this pattern. The spectacular technologic advances in mining methods have been made at large mines. Given a suitable mill to treat mixed ores, however, the workings even in small mines can be made wider, and less care need be taken to avoid dilution of ore in laying out stopes. Even in small and spotty deposits where the workings must still be narrow and crooked, the use of sharper drill bits, more efficient rock drills, portable compressors, and better pumps and hoists will help to offset higher wages and other costs. It remains to be seen how great the savings will be.

In the days when efficient labor was very cheap in the low wage areas where so many of the pegmatite mines were situated, it was notoriously true that even the best mines were worked only intermittently by a succession of different operators. The same dike might be worked exclusively for one mineral for a time and then, after a lapse of months or years, it might be worked for another. Fluctuating markets, lack of capital, and above all the seemingly unpredictable nature of these baffling deposits have discouraged attempts at systematic mining. Temporary abandonment may not cause cave-ins that ruin the orebody, but a good many deposits have been ruined by gophering. Absentee owners have been plagued by unauthorized working of their mines, pilfering of tools, and sabotage to buildings. At some operations high-grading is a problem. With the government paying up to $70 per lb for mica there is a strong temptation to establish a black market for mica bootlegged from legitimate operations. The magnitude of this problem is best illustrated by Indian official statistics which revealed that as much as 2.4 lb of mica were stolen or illegitimately produced for every pound marketed through regular channels.

The most serious threat to the establishment of a successful pegmatite industry that will supply strategic minerals from domestic mines is the mounting difficulty of operating small mines. The latest census data relate to 1939, and the situation has improved very little since then. In that year only four feldspar companies marketed over $50,000 worth of products. Many feldspar, mica, and other pegmatite mining enterprises were too small to be counted under the census rule, which excluded those producing less than $2500 worth of products or spending less than $2500 in their business. The average feldspar miner in North Carolina produced only 0.156 short tons of crude spar for a wage of $0.28 per hr, and in Colorado the highest state average was only 0.415 tons per man-hour for a wage of $0.53. For all 59 feldspar mines reporting to the census in that pre-war year, the average value of products per man-hour was only $0.96; for mica mines it was $0.91. These figures compared with $3.95 at iron mines and $2.76 at copper mines.

By organizing a group of mining units around a central mill and by utilizing old dumps to augment the tonnage from current mining, it may be possible in several districts to develop pegmatite mineral production units of economical size even where no really large deposits can be found. Even with these additions, however, the probable number of larger pegmatite operations is limited and the extent to which they can be depended upon to supply rarer elements as byproducts is debatable. The fact remains that as wages and labor conditions rise to meet higher standards throughout the world, the sources of supply of any mineral that comes principally from isolated small and highly irregular pegmatite deposits become more and more vulnerable. In terms of national policy, this calls for even greater emphasis upon the search for substitutes or ways and means of getting along without certain of these minerals, of which mica and beryl seem to be especially critical.

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St. Joseph Lead's Indian Creek Development

by C. Kremer Bain

DURING the past several years of diamond drilling in Washington County, Mo., the St. Joseph Lead Co. has discovered a concentration of commercial lead-zinc ore at four different points within an area of 36 square miles. The largest and to date the most promising group of these orebodies is located in the Indian Creek area. It was considered to be of sufficient value to warrant development.

The Indian Creek area is in the center of the northern half of Washington County, 11 miles northwest of Potosi, Mo., the county seat. The location is readily accessible from Potosi by car or truck over the all-weather Missouri State Highway No. 114.

The topography is one of rolling relief, varying from an elevation of 700 ft above sea level on Mineral Fork and Indian Creek to a maximum of 1176 ft at High Point Tower. It is divided in equal halves between the drainage to Mineral Fork and to Indian Creek. The shaft site at 1060 elevation is on the divide between the two drainage areas. The Indian Creek side is rough and heavily wooded, with only a few cultivated clearings. The slopes to Mineral Fork are more gentle and rounded, with much of the area cleared for farm land. Most of the surface is covered with Potosi and Eminence formations, which show little evidence of any structure. Ore structure was worked out by extensive diamond core drilling, with holes varying in depth from about 700 to 1300 ft. As of Oct. 1, 1950, 218 prospect holes had been drilled within the 36-square mile area.

The ore is disseminated galena with considerable accompanying sphalerite in the Bonne Terre formation in the shape of irregular blanket-type deposits. The main controlling structural feature of the outlined mineral zone is a buried pre-Cambrian rhyolite-porphyry ridge extending partly through the Bonne Terre formation, which laps against the porphyry and also covers it. The mineralized zone parallels and forms an irregular halo around the porphyry ridge and is concentrated in the Bonne Terre formation where the La Motte sandstone is cut out by the porphyry ridge. This is a very common mineral occurrence in the southeastern Missouri lead belt district. The La Motte sandstone, which normally underlies the Bonne Terre formation, is thus cut out entirely. In this area the Bonne Terre is usually about 275 to 320 ft thick, the La Motte varying in thickness from zero up to 400 ft.

Commercial concentrations of ore are indicated at several places, the most notable being at the shaft site where an orebody at least 4000 ft long with widths of 500 to 600 ft has been outlined. The total thickness of mineralized rock reaches 150 ft in some holes, although total ore thickness runs from 8 ft to about 60 ft, often in more than one layer.

Preliminary Planning

Ore reserves as outlined lend themselves to open-stope mining with trackless haulage and mobile loading machines and drilling equipment. This type of equipment has proved to be very efficient during recent years in the company's lead belt operations.

After a study of bids for shaft sinking and an estimation of comparative costs, it was decided that the shaft would be sunk by St. Joseph Lead Co. personnel using company equipment.

Since the plant site is about 26 miles from the company's nearest lead belt operations, it was necessary to plan a complete surface plant, including mill, shops, office, change room, and warehousing facilities.

Immediately following the filing of an application for a certificate of necessity, a headframe was selected for which detailed drawings were already available. The order was placed for fabrication so that it could be delivered and erected in time to serve for sinking the shaft. This eliminated the expense of a temporary headframe.

Drilling was started on two churn drill holes near shaft location, and two 1000-gpm deep well pumps

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Discussion on this paper, TP 3601A, may be sent (2 copies) to AIME before Nov. 30, 1953. Manuscript, Jan. 15, 1953. Los Angeles Meeting, February 1953.

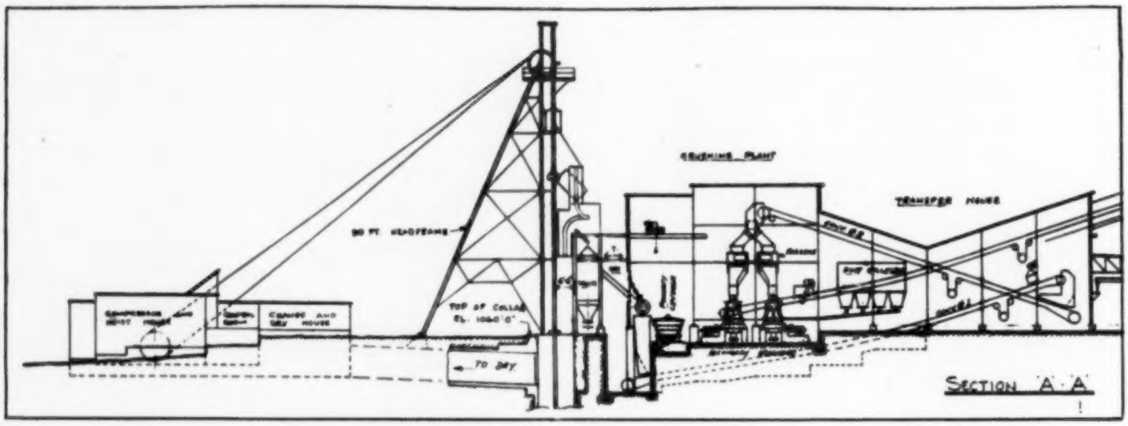


Fig. 1 (above)—The general arrangement of equipment at Indian Creek. Fig. 2 (right)—A plan of the surface layout at Indian Creek.

were ordered for installation in these holes in an attempt to lower the water table ahead of the sinking operations.

Contracts were drawn with Arkansas-Missouri Power Co. for construction of 11 miles of 33,000-v power line from Potosi and for installation of a bank of three 1000-kva, 33,000/440-v transformers at the plant site. Dependable and quick communication being secondary only to power on a project of this size, arrangements were also made for a telephone line from Potosi to the plant site.

Company geologists selected a lead belt stope from which ore was being mined similar to that indicated by the drill cores at Indian Creek. A sizeable sample was taken from this stope for metallurgical and grinding tests to determine the flowsheet to be used in the new mill of 2000 tons per day capacity.

The nearest railroad to the plant is a spur of the Missouri Pacific which runs from Mineral Point, Mo., to Potosi. Topography of the intervening country makes it impracticable to extend a rail connection to the plant. However, the all-weather graveled highway into Potosi should be adequate for heavy truck transportation. Lead and zinc concentrates will be hauled by truck to Potosi where they will be loaded into cars at the rail head for shipment to the smelter. Operating equipment and supplies will be trucked from Potosi or from the St. Joseph Lead Co. central warehouse located in the lead belt about 30 miles from the plant site.

The topography of the plant site selected permitted location of the shaft very close to the lowest elevation of ore and provided a hillside slope for the concentrator as well as gravity flow for the tailings disposal. Thus fewer pumps will be required. A reservoir located 400 ft from the mill site will furnish water storage at a static head of 80 ft, eliminating the need for mill feed pumps.

Standardization of equipment to conform with newest types of company installations in the lead belt was necessitated by the isolated location of the plant and need for interchangeable spare parts.

Requirements for separation of lead and zinc concentrates called for a differential flotation plant. To minimize equipment requirements and still maintain the advantage of separating free lead in the coarser feed which, in the lead belt, is recovered by gravity concentrating tables, it was decided to use jigs for scalping between the relatively coarse grinding rod mill, and the classifiers in closed circuit with the ball mills.

A contract was awarded to General Engineering Co., Ltd. of Toronto to design the surface plant and supervise construction. Construction work was subcontracted under supervision of a project engineer and construction superintendent from General Engineering Co. who worked on site. Sub-contracts were given with the provision that the contractor would use local labor whenever possible, so that a favorable company-community relationship would be established. Fabrication and erection of steelwork, as well as plumbing, heating, and electrical work, were given to established St. Louis firms. All tile work was awarded to the local contractor nearest the Potosi area.

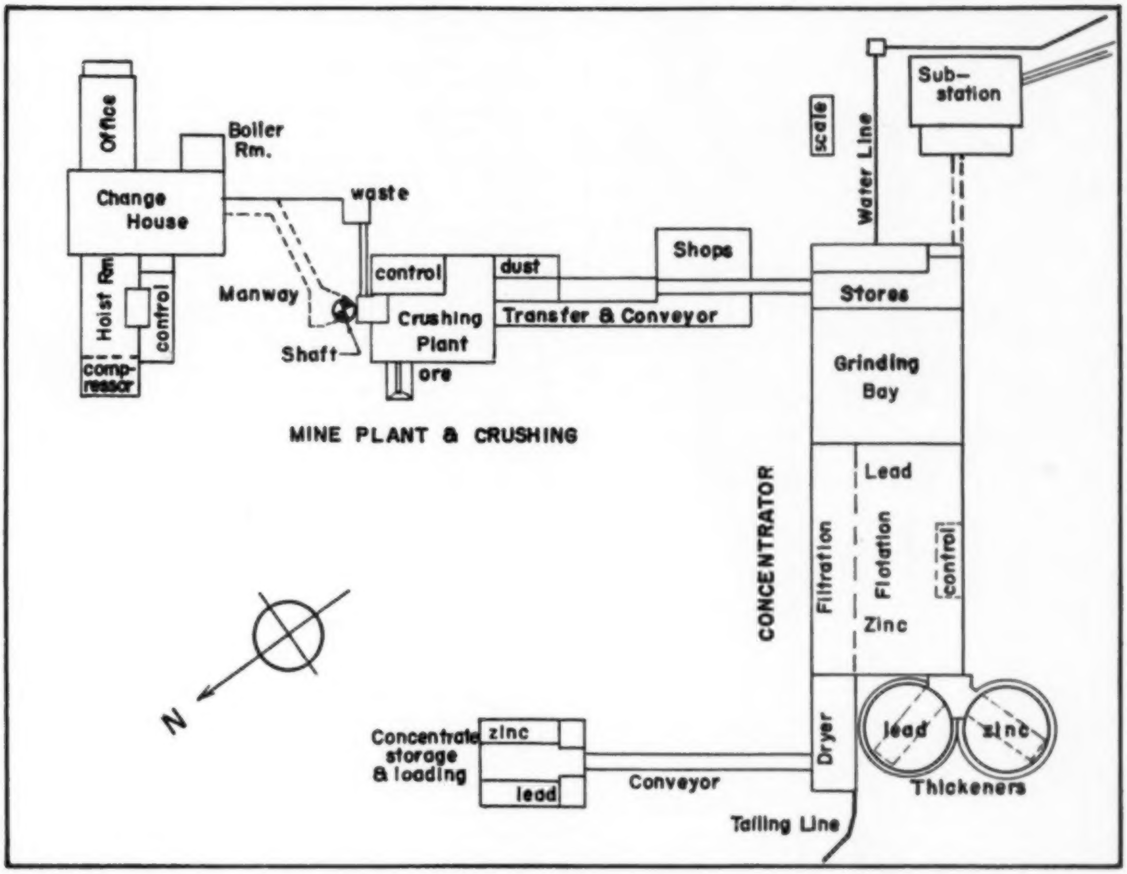
Plant Layout

The frontispiece is an artist's conception of the plant. Figs. 1 and 2 are the surface layout, with location of the two-story office building, change house, heating plant, hoist and compressor room in relation to the shaft, crushing plant, concentrator, concentrate loading stations, sub-station, and reservoir. It is to be noted that the supply warehouse is located immediately under the ore bin, and surface shops under the conveyor gallery in what ordinarily would be wasted space. Thus shop service and supplies are in the center of operations. All buildings are of fireproof construction, with steel framing and Spectra-Glazed tile walls. The tile is a haydite block with ceramic paint baked on one or more surfaces. It can be ordered with different colors on different faces of the tile, so that outside and inside finishes of the building are automatically furnished.

Many modern features were incorporated in the new change room. Outstanding among these are the radiant heating system in the floors and the tunnel connection to a point 20 ft below the shaft collar, making it possible for men to go in and out of the mine without being exposed to the weather. Lockers for street clothes are recessed in the walls, and there are hanging baskets for work clothes. These baskets may be raised to the ceiling by cords attached to individual lockers. The floor is finished with antibacterial cement for prevention of athlete's foot.

Hoist

The hoist will have 1⅝-in. diam ropes overwound and underwound on a 10-ft diam single drum with 8-ton skips hoisting in counter-balance at a rope



speed of 816 fpm. It will be powered by a 600-hp, 2300-v motor and will have a capacity of 350 tons per hr when hoisting from the 920-ft depth.

The loading arrangement at the bottom of the 1500-ton capacity skip pocket is believed to be the first of its type to be used in non-ferrous metal mining operations. Fig. 3 shows a cross section of the general arrangement of the skip pocket and loading equipment. The pan feeder will draw rock from the pocket and discharge into a hopper where the rock will be weighed by an electronic device to the tonnage decided upon for each skip. The operating console will be arranged so that skip loading can be manual or automatic.

Power

Power will be supplied by the Arkansas-Missouri Power Co., which in turn receives power from a Union Electric Co. sub-station at Shirley, Mo. This new 10-mile power line from the sub-station connects with the St. Joseph Lead Co. transformer station at plant site. The transformer station will be an outdoor packaged unit complete with structure, bus, wire, hardware, primary air switches, fuses, lightning arresters, 7500-kva, 33,000/2300-v transformer, and secondary breakers. It will furnish 2300-v power to the shaft and to distribution centers in the hoist room, crushing plant, grinding section, and flotation section. All motors over 150-hp will be 2300-v and smaller motors will be 440-v. The 2300/440-v transformer will be located at or near load centers. Distribution will be from central metal-clad power control centers with pull-out type, removable circuit breakers.

All control centers will be located in an air-conditioned dust-free room, and all motors will be controlled by push buttons located near the operator's position, with interlocking controls where necessary to insure proper sequence of starting and stopping. E. H. Tucker of Newmont Mining Corp. designed the electrical layout.

Mill

The graphic flowsheet, Fig. 4, shows size and type of crushing, grinding, and flotation equipment to be used for handling 2000 tons per day. The crushing section has a capacity of 2000 tons in 6½ hr, the same as that of the hoisting equipment. Storage bin facilities of 2200 tons will insure ample feed to the grinding and flotation section for the 2000 tons per 24 hr. If future conditions warrant expansion of this mill, to double plant capacity it will only be necessary to duplicate the grinding and flotation section and go to two shifts hoisting and crushing.

Shaft Sinking

It was decided that a circular shaft with 12-in. thick concrete wall would give the strongest section through the upper Potosi formation, which showed every indication of being heavily water-bearing. The shaft also would provide maximum clearance with the least amount of excavation to accommodate hoisting and lowering of large trackless haulage equipment. An ID of 12 ft 7 in. would allow for handling of 8-ton skips plus thickness of guides and supports.

Years of experience in operations have proved that when machines are simply and properly designed even inexperienced crews, under supervision,

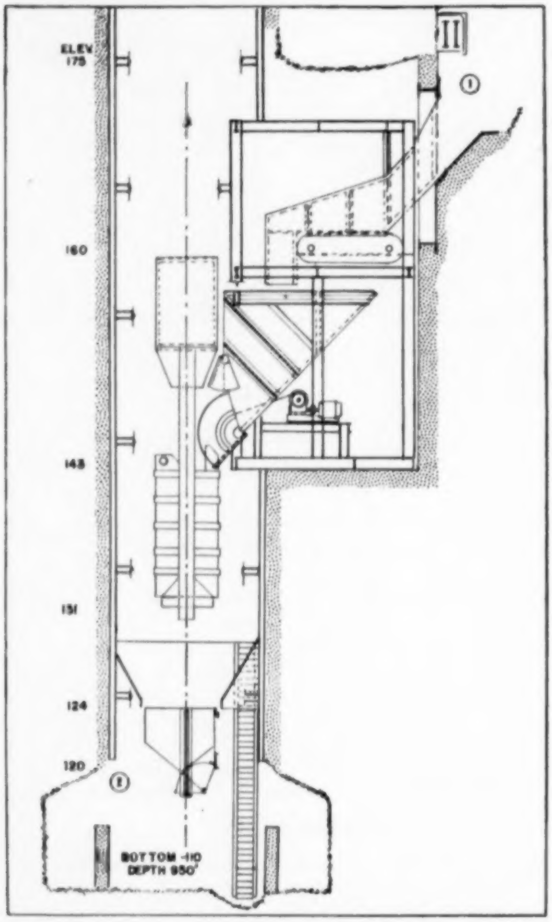


Fig. 3—Arrangement of the skip pocket. Circled numeral 1 indicates 1500-ton capacity skip pocket. Numeral 2 indicates point at which shaft concreting stops, 940 ft from the top of the shaft.

can sink a shaft safely and efficiently. A number of pieces of equipment were designed specifically for sinking the Indian Creek shaft. When possible, plans were obtained for still other pieces of equipment found suitable on similar shaft-sinking operations in other districts.

Final plans called for sinking of a cylindrical shaft 950 ft in depth and about 15 ft in diam. With the exception of landing stages and the bottom 10 ft, the entire shaft was lined with concrete to an ID of 12 ft 7 in. One 10-in. pipe and four 4-in. pipes were placed in the concrete lining in one quadrant. Thus located in the concrete lining, these pipes would not only be out of the way during future hoisting operations but also could be used in the actual sinking operations for compressed air, ventilation, pump discharge, and many other purposes.

A circular, four-sectioned steel form was designed for use in pouring this concrete lining in increments from 5 to 20 ft. This steel form remained in place on the last segment of lining poured until sufficient shaft was sunk for another section. A winch and drum containing 1000 ft of ⅝-in. steel cable was installed on a concrete pier at surface and a sheave installed on the inner lip of the shaft collar for each of the four sections of the steel concrete forms. By this arrangement, the forms were easily lowered into position for pouring the next segment of the shaft lining.

Convenient, yet out of the way, these steel concrete forms protected the green concrete lining from both fragments and blast effect. Sufficient drill holes and powder were used to insure relatively fine fragmentation to reduce further the possibilities of damage to the shaft lining and other installations. Concrete lining was carried as close as possible behind the sinking crew to protect the men from any possible rock falls or cave-ins and to minimize scaling. In addition, ground water was sealed off as sinking operations advanced.

A column of 6-in. H-beams, located at each of the four quadrant points, was designed to serve as bracing for the steel concrete forms and to act as guides for the work cage and other pieces of equipment. After completion of the shaft-sinking operations, these H-beams will serve as supports for actual skip guides. They were prefabricated, drilled to a template, and anchored by ¾-in. hook bolts set in the concrete lining. A template, *wing forms*, and *fan tail* jacks were used to locate the H-beams in position where they could be jacked solid after being plumbed.

A double-decked steel work cage was designed for shaft-sinking operations, with guide shoes engaging each of the four H-beam columns. This cage not only served as a platform from which the crew could work at any point in the shaft, but also provided facilities for lowering equipment and materials, and acted as internal bracing for the forms as the concrete was being placed.

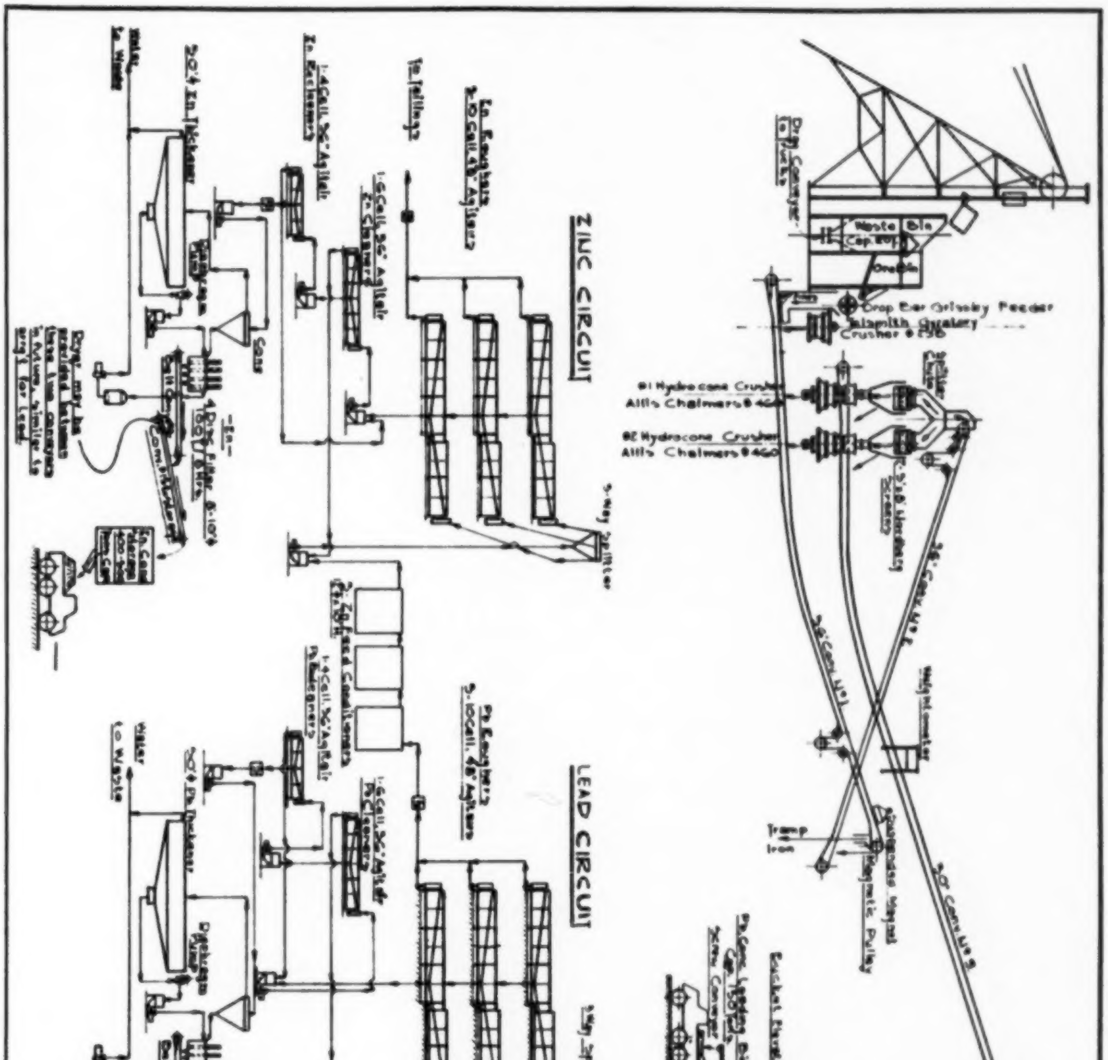
All electrical control wires were suspended from a steel cable and connected to a canopy. These included wires for lighting systems, speaker system, signal system, electric hoist controls on the work cage, and mucking machine controls. Outlet plugs for various control wires, a speaker, and lights were installed on this canopy.

A positive displacement Roots-Connersville blower, designed to supply air at 4000 cu ft per min at 2½ psi, was connected to the 10-in. pipe set in the concrete lining. This blower furnishes ventilation for the shaft and subsequent operations until permanent ventilation is established.

Also used in sinking operations were customary items of equipment such as grout and sump pumps, compressors, drill presses, electric and acetylene welding sets, and a concrete batching plant to insure a proper and uniform mix. Concrete was delivered from the mixer to a 1½ cu yd bottom dump concrete bucket by means of a 24-in. trough conveyor. Washed sand and gravel were stockpiled near the batcher. Water was supplied by a deep well pump, and cement was stored under shelter convenient to the batcher.

A suitable drill round with all holes properly located was of great importance. Even the most experienced drill crews often failed to spot and drill holes of a round accurately. This is considered to be a major factor contributing to misbreaks. A four-machine jumbo drill was therefore designed for the Indian Creek shaft. Each machine was independent of the others. A fixed angle between the feed and the arms, in conjunction with a special indexing device, provided a rapid and accurate

Fig. 4—The graphic flowsheet on the righthand page lists the size and type of equipment in use at the St. Joseph Lead Co. Indian Creek development.


ZINC CIRCUIT
LEAD CIRCUIT
MILL BIN
Capacity 2700 Tons
Waste Bin
Ore Bin
Drop Bar Grizzly Feeder
#1 Hydrocone Crusher
Allis Chalmers #460
#2 Hydrocone Crusher
Allis Chalmers #460
Tramp Iron
Bucket Elevator
Tripper
To Tailings

method of aligning drill steel and spotting the hole with proper pitch and depth.

A 56-hole round of drill holes, each 1⅝-in. in diam, was used for breaking the shaft. By means of the special features on the jumbo, the drills could be set for breaking a theoretical 8, 9, or 10-ft round. This particular shaft round might be classed as a modified conical cut with two rings of relief and one ring of line holes. By means of the jumbo, all holes could be drilled accurately with cuts actually meeting at a point, and line holes could be drilled to minimize overbreak.

Mucking the round in a shaft has generally been considered one of the toughest and most time-consuming jobs in sinking operations. Clamshells had been used successfully on other mucking jobs, so a mucking machine with the clamshell powered by two air motors with spring-loaded brakes was designed for this operation. Motors were located on the surface and operated by remote control from the bottom of the shaft. By this means, the operator had complete control of the clamshell from a position where he could closely observe its operation. The muck was hoisted in a 54-cu ft bucket and dumped by means of a hinged dumping door, while the shaft men were protected by a sliding shaft door. A scraper was used to drag waste rock from the vicinity of the shaft and dispose it in a nearby ravine. Plans for these doors and bucket were obtained through courtesy of the Tennessee Copper Co., Ducktown, Tenn.

The shaft-sinking crew consisted of a general foreman, one shaft foreman, one hoistman, one maintenance man, and four shaft men on each of three shifts. Three shaft men on each crew were recruited from the vicinity and had had no previous experience in mining. None was afraid of work and all were willing to learn. The operation and functioning of the equipment were new to all members of the crew, since it was the first time that much of it had ever been assembled on any job.

Weekly inspections were made of ropes, sheaves, hoist, and compressors, by a machinist from the shop organization in the lead belt. The regular shaft crew handled all maintenance work and breakdowns.

Summary

It is worthwhile to point out that results of company planning were satisfactory both as to cost and progress. Sinking was started with an inexperienced crew, a number of whom had never been underground. The shaft was bottomed by a smooth-working team of experienced shaft men. Their outstanding record of completing the entire 950-ft depth, averaging 7 ft 4 in. per blast without a single misbreak, with excellent fragmentation, and without a single lost time accident speaks for itself.

Technical Note

Calcined Cold-Precipitated Hydrated Iron Oxide

by William A. Mitchell

AN X-ray diffraction pattern for "calcined cold precipitated ferric oxide" is reproduced diagrammatically along with data for other iron oxides by R. C. Mackenzie.[1] This pattern, which shows spacings higher than those for any other form of iron oxide, was obtained from material prepared and photographed at several different times over a period of weeks, but later attempts to reproduce the results were unsuccessful, the pattern of hematite being obtained every time.[2]

In an attempt to solve this problem, the original photographs showing the strange diffraction pattern were examined and a marked similarity to the hematite pattern was noticed, suggesting that the pattern was in fact due to hematite but corresponded to a shorter wavelength of X-rays than the Cu Kα which was assumed to have been used.

The eight strongest lines were provisionally identified with the strongest hematite lines and, by means of the corresponding hematite spacings, the X-ray wavelength was calculated for each line. This gave an almost constant value, with the exception of the innermost line which was somewhat diffuse, and confirmed the presence of hematite. The value obtained, 0.703 ± 0.015A, corresponds to the molybdenum Kα wavelength.

W. A. MITCHELL is a member of the Macaulay Institute for Soil Research, Aberdeen, Scotland. TN 176H. Manuscript, May 20, 1953.

The X-ray tube used was a four-filament demountable tube with a copper target. Other photographs, mostly of soil clays, taken during the same period gave normal patterns of Cu Kα radiation with one exception. This was a calcined soil clay with a high iron content and also showed Mo Kα lines of the hematite structure.

The explanation is almost certainly that the copper target was contaminated with molybdenum, but it is not known how this occurred. Specimens of alumino-silicates gave their normal Cu Kα pattern, but as the K absorption edge of iron is a little longer than the Cu Kα wavelength, specimens with a high iron content greatly reduced the intensity of the Cu Kα diffraction lines relative to that of the lines due to the shorter Mo Kα wavelength.

The formation of hematite on calcining cold precipitated hydrated ferric oxide[3] is thus confirmed.

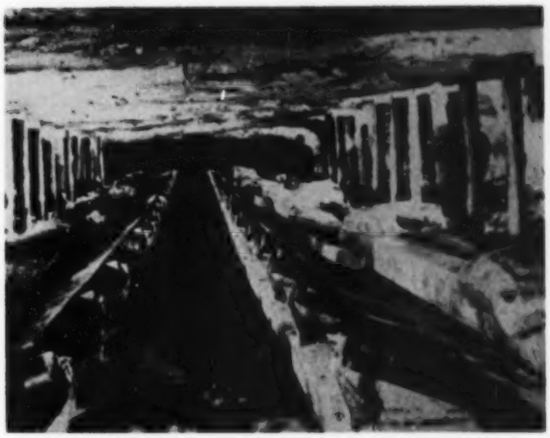
References

[1] R. C. Mackenzie: Investigations on Cold-precipitated Hydrated Ferric Oxide and its Origin in Clays. Problems of Clay and Laterite Genesis, p. 69, Fig. 4, F. AIME Symposium, St. Louis, Mo. Feb. 19-22, 1951. Published 1952.

[2] R. C. Mackenzie: Ref. 1. Note added on proof.

[3] W. C. Hansen and L. T. Brownmiller: Equilibrium Studies on Alumina and Ferric Oxide and on Combinations of These with Magnesia and Calcium Oxide. *American Journal of Science* (1928) 5, No. 15, pp. 225-242.

Two-Way Belt Conveyor Transportation



by C. W. Thompson

Fig. 1—Simultaneous haulage of coal and supplies.

The two-way belt conveyor for coal mine service simultaneously carries coal from faces and transports men and supplies into the mine, largely eliminating the necessity for rubber-tired supply and man haulage equipment and the necessity of maintaining additional headings.

SINCE 1920 use of belt conveyor transportation has increased steadily, and capacity in tons per hour per unit has increased at about the same ratio. Belt conveyor systems for simultaneous two-way material transportation have been widely discussed, but heretofore have proved impractical for specific operations.

Studies of a proposed means of disposing of washery waste on the return trip of a main haulage conveyor, as yet uninstalled, led to the application of two-way belt haulage to Weirton's underground supply and mantrip problems.

Except for a single-tracked main haulage entry, Weirton mine is essentially a full-belt haulage operation with attendant problems of supply and mantrip haulage that could not be handled with conventional belt conveyors during the regular work shift. It was necessary, therefore, to maintain rubber-tired haulage equipment and haulageways serving multiple-shift continuous operations.

The conventional underground belt conveyor is primarily intended to transport coal from working faces to surface. Belts are usually reversed to haul men to face areas at the start of the working shift or to transport supplies.

The two-way belt conveyor for coal mine service is not yet offered for sale as a unit by any manufacturer and must be built on order. This conveyor, with return run traveling parallel but opposite in direction to the outgoing or coal haulage run, permits not only haulage of coal from faces, but simultaneously permits transportation of men and supplies into the mine at any time, see Figs. 1 and 2. This eliminates, to a great extent, the necessity for rubber-tired supply and man haulage equipment and the maintenance of additional headings. Unlike conventional installations the return belt is in the open for inspection and maintenance. It is less subject to edge wear, since it runs on regular troughing idlers, see Fig. 3. The only points where the edge of the belt can rub are at the drive unit and the tail units where improved design should eventually eliminate edge wear entirely.

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Individual conveyor sections are lighter, since all that is needed is a tubular stringer on each side to carry troughing idlers only, no cover plates being necessary. The return run is carried on an assembly identical to that carrying the coal haulage run. These assemblies are definitely easier to handle and are cheaper than installations for the same length of conventional belt conveyor.

The four pipes required for the complete conveyor frame may serve only as supports, or, with proper couplings, may be used for mine water discharge line, pressure line for face spraying, telephone cable conduit, or any other purpose requiring a pipe line, see Fig. 4.

Additional expense, if any, for the increased number of pulleys needed to turn the belt over for the return service is more than offset by reduced expense for rubber-tired supply and mantrip haulage.

The two-way belt haulage system is confined to a single heading on intake air, isolated from all other headings by stoppings on each side, and open for air circulation only at the main haulage way and at the faces. One additional heading, also on intake air, is maintained as an emergency rubber-tired haulage way, all other headings being center-posted and used as airways only until retreat work begins. This method, of course, can and should be used with any belt conveyor, since it provides two independent intake airways for escape in case of fire on the belt or elsewhere. Belt heading is heavily timbered even under good top conditions and space between the two runs permits center posting for the full length

Fig. 2—Mantrip and simultaneous haulage of coal.

of the conveyor system. Top or bottom is taken when necessary at the points where vertical pulleys required for the quarter-turn twists are to be installed. The units where the twist is made are collapsible, Fig. 5, so that they may be moved under low top when the belt is extended. The twist units are installed in the same places on retreat as on advance so that further top or bottom brushing is unnecessary.

Each new installation incorporates improvements brought about by previous experience. The following data from a manufacturer's development problem report presents methods of determining the distance needed for a quarter-turn twist, Fig. 6, a major consideration in the design of the two-way belt conveyor.

The distance required to make the quarter-turn twist will vary as the duck weight, unit tension, and belt width. This report covers 28-ounce, 32-ounce, 42-ounce, and 48-ounce duck with tensions and widths which seem to be practical for a two-way conveyor.

The chief problem in determining distance for the quarter-turn twist was to find a modulus of elasticity for the various ducks. Tests previously made on these ducks were reviewed and stress-strain charts were plotted. Two curves were drawn for each duck weight, these curves being the extremes of the scatter band. The high curve on each chart is a high modulus of elasticity which, in the case of the two-way conveyor, is the most undesirable and the modulus of interest. For the purpose of discovering whether or not a belt used in a quarter-turn twist would distort as calculated from the modulus of elasticity, an 18-in., 4-ply, 28-ounce belt was set up in the laboratory, Fig. 7. Increments of 23 in. were measured off with no tension; then a known tension was put on the belt and the modulus of elasticity was calculated. This modulus was found to fall below the scatter band of the stress-strain curves made from older tests.

Another problem was to determine whether or not the belt between the horizontal pulley and vertical pulley absorbed the stretch required to make the quarter turn, or whether the belt relayed some of the elongation past the pulleys until some unknown length absorbed it. From the quarter-turn drive set up in the laboratory, it was found by measurement that this elongation must be absorbed between the horizontal and vertical pulleys. This test also proved that minimum tension was at the belt center, increasing on a straight line to a maximum tension at the edge. By means of modulus of elasticity, belt widths, and various belt tensions, calculations were worked out to determine distance required for the necessary quarter-turn twist for a two-way conveyor.

The following method is used to calculate distance required for the quarter turn, see Figs. 8 and 9.

Ts = Tension at edge of belt in the quarter turn drive, lb per ply in. Edge of belt travels helical path in making quarter turn.

Tm = Tension at center of belt, lb per ply in.

T = Total tension on belt, lb.

L = Center to center distance of vertical and horizontal pulleys, in.

Σ = Change in unit elongation from edge of belt to center of belt, in.

ϵ = Total elongation of edge with respect to center, in.

$\Delta T = Ts - Tm$

Fig. 3—A view of the experimental underground two-way conveyor.

E = Modulus of elasticity

W = Width of belt, in.

$$WTm + \frac{W}{2}(Ts - Tm) = \frac{T}{\text{No. of Plies}} \quad [1]$$

Fig. 9 is a linear presentation of Eq. 1.

T is known from requirements of the belt. Tm has a minimum of O, and Ts a maximum depending on the weight of duck used. The maximum allowable tension for various duck weights is as follows: 28 ounce, 34 lb per ply in.; 32 ounce, 39 lb per ply in.; 42 ounce, 49 lb per ply in.; 48 ounce, 59 lb per ply in.

$$\Sigma = \frac{\Delta T}{E} \quad [2]$$

Since the path of any point on the edge of the belt in the quarter turn is a helix, the solution can be found by the triangular method whereby the helix is the hypotenuse of the triangle, the distance between pulleys the longer side, and the pitch and shorter side one half the belt width.

$$L + \epsilon = \sqrt{L^2 + \left(\frac{\pi W}{4}\right)^2} \quad \text{(For 90° turn)}$$

$$\epsilon = \Sigma L$$

By substitutions and simplifying:

$$L = \frac{\pi W}{4}\sqrt{\frac{1}{\Sigma(2+\Sigma)}} = 0.785W\sqrt{\frac{1}{\Sigma(2+\Sigma)}} \quad [3]$$

The above solution can be used only when the quarter turn is symmetrical as in Fig. 8. Fig. 9 shows a quarter-turn drive that is not symmetrical, and with this drive the following equation must be used. See also Figs. 10 and 11.

T_{SL} = Tension in the low tension edge of the belt in the quarter turn drive, lb per ply in.

T_{SH} = Tension in the high tension edge of the belt, lb per ply in.

T_{MIN} = Tension in the minimum tension section of the belt, lb per ply in.

L = Center to center distance of vertical and horizontal pulleys, in.

y = Distance from high tension edge to minimum tension section, in.

x = Distance from low tension edge to minimum tension section, in.

ΣH = Change in unit elongation from high tension edge to minimum tension section.

ΣL = Change in unit elongation from low tension edge to minimum tension section.

$\Delta T_H = T_{SH} - T_{MIN}$

$\Delta T_L = T_{SL} - T_{MIN}$

E = Modulus of elasticity.

Fig. 4—Two of the four pipes supporting the conveyor frame are shown here. One pipe carries water in the mine for spraying, another pipe carries mine water out of the mine, the third contains air for roof bolting, and the fourth carries telephone line.

Fig. 5—Here there is a turnover in 46-in. coal without the necessity of taking any roof.

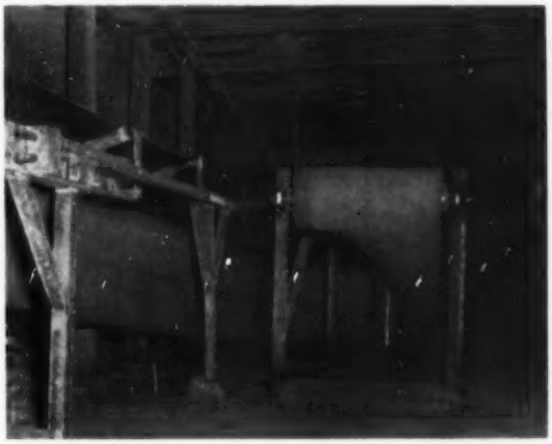

Fig. 6—The quarter-turn twist on the experimental underground unit.

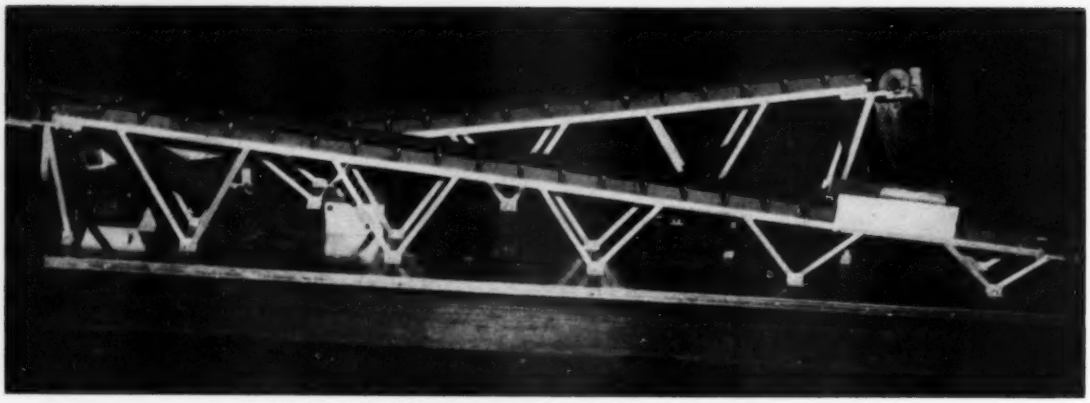

Fig. 7—The experimental unit in the mine administration building.

W = Width of belt, in.

T = Total tension of the belt, lb.

$$WT_{MIN} + x\frac{(T_{SL} - T_{MIN})}{2} + y\frac{(T_{SH} - T_{MIN})}{2} = \frac{T}{\text{No. of plies}} \quad [1\text{-A}]$$

$$\frac{y}{T_{SH} - T_{MIN}} = \frac{x}{T_{SL} - T_{MIN}}$$

$$T_{SL} = \frac{x\,(T_{SH} - T_{MIN}) + y\,T_{MIN}}{y} \quad [1\text{-B}]$$

Substitute this value for T_{SL} in Eq. 1-A. The result is Eq. 1-C.

$$WT_{MIN} + \frac{(x^2 + y^2)\,(T_{SH} - T_{MIN})}{2y} = \frac{T}{\text{No. of plies}} \quad [1\text{-C}]$$

By using Eq. 1-C it is possible to determine T_{SH} and T_{MIN}. First substitute allowable duck strength in pounds per ply inch for T_{SH} and solve for T_{MIN}. If T_{MIN} is 0 or greater, then the values of T_{SH} and T_{MIN} are satisfactory. However, if T_{MIN} is less than 0, it is then necessary to set $T_{MIN} = 0$, and solve for T_{SH}. In determining L, T_{SH} and T_{MIN} are critical factors. This may be stated because T_{SL} will never be greater than T_{SH} nor will it be less than T_{MIN}. In each case, the modulus of elasticity used is the high modulus as determined from the stress-strain charts. It is felt that the modulus of elasticity used in making the calculations for Table I will be sufficiently high for conveyor belts now in factory production.

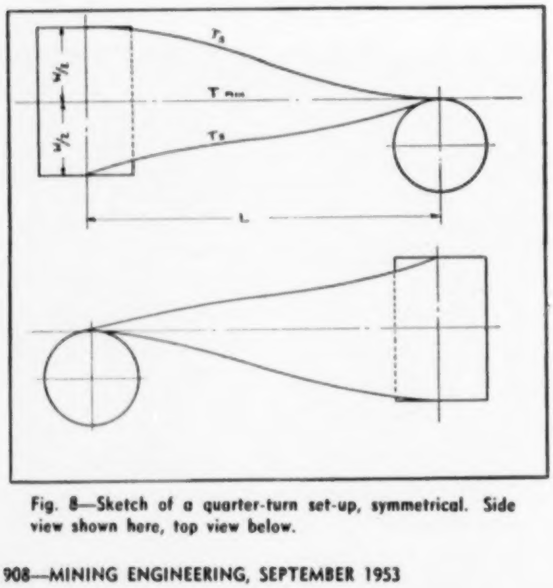

Fig. 8—Sketch of a quarter-turn set-up, symmetrical. Side view shown here, top view below.

$$\Sigma = \frac{\Delta T_H}{E}$$

$$L + \epsilon = \sqrt{L^2 + \left(\frac{\pi y}{4}\right)^2} \quad [2\text{-A}]$$

$$\epsilon = \Sigma L$$

By substitution and simplifying:

$$L = \frac{\pi}{y}\sqrt{\frac{y^2}{2\Sigma + \Sigma^2}} = 0.785y\,\sqrt{\frac{1}{\Sigma(2+\Sigma)}} \quad [3\text{-A}]$$

Table I. Distances Required For Twisting Belts

	Width, In.	Tension, Lb Per Ply In.*				
28 Ounce Duck		10	15	20	25	20
	18	126**	102	106	133	199
	24	167	136	141	177	265
	30	209	171	177	221	331
	36	251	204	212	265	398
	42	292	239	248	309	464
	60	418	341	353	442	663
32 Ounce Duck		15	20	25	30	35
	18	96	86	100	125	187
	24	129	114	133	166	250
	30	161	143	168	208	312
	36	193	171	200	249	374
	42	225	200	234	291	437
	48	257	228	266	332	500
	60	321	286	333	415	624
42 Ounce Duck		20	25	30	35	40
	18	92	84	95	110	138
	24	123	112	126	147	183
	30	153	140	157	184	239
	36	184	168	189	220	275
	42	214	196	220	257	311
	48	245	224	252	294	367
	60	306	280	315	367	458
48 Ounce Duck		25	30	35	40	45
	18	87	81	89	100	117
	24	116	108	119	133	155
	30	145	135	148	167	194
	36	174	162	178	200	233
	42	203	189	208	234	272
	48	232	216	237	266	310
	60	290	270	296	333	388

* Unit tension in straight portion of the belt approaching the drive.

** An upturn in length of twist at low tension is caused by the need of avoiding compression in the center of the belt, rather than by high edge tension.

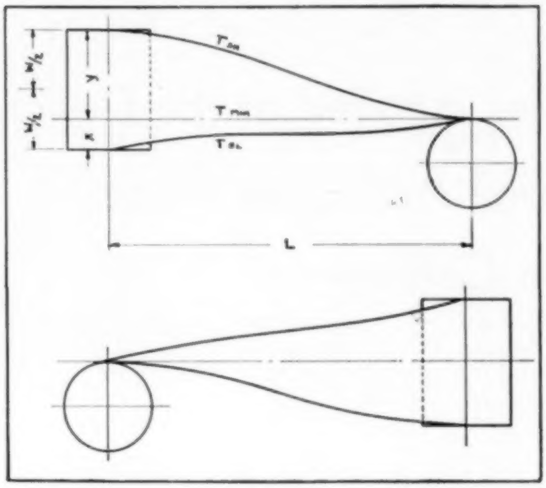

Fig. 9—Sketch of quarter-turn set-up, not symmetrical. Side view shown above, top view below.

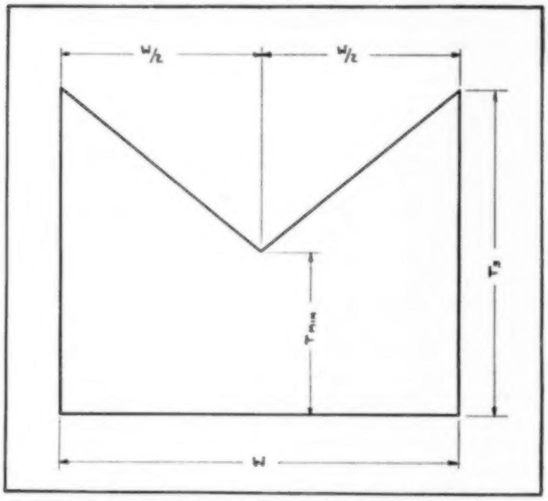

Fig. 10—Tension distribution across width of belt for a quarter-turn drive, symmetrical.

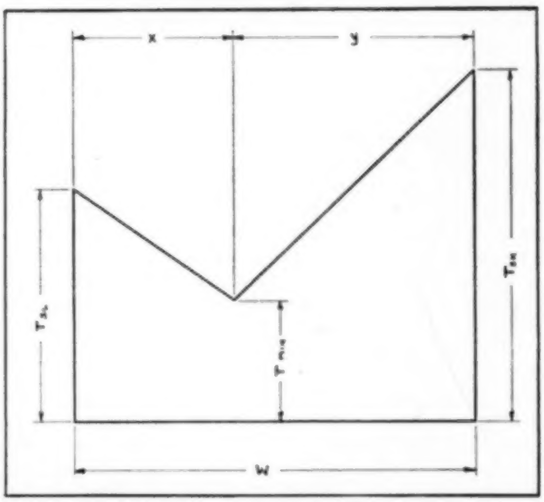

Fig. 11—Tension distribution across width of belt, not symmetrical.

Table I shows the distance required to make the quarter turn for various duck weights, belt widths, and belt tensions.

It is apparent that the twist required of the belt in the two-way conveyor can be made in a distance that is entirely reasonable. The only other problem, from the belt standpoint, is that of training the belt. This is shown to be practical by the laboratory set-up and by existing quarter-turn drives. From the standpoint of equipment there will be, in some cases, a problem of sufficient head room to stand pulleys on end for the twist in the belt and a problem providing additional structure to support the return belt. Experience at Weirton mine has proved that the problems are not difficult to solve.

Belt manufacturers' calculations show the two-way belt to be practical for lengths up to one mile with ordinary belting. For greater lengths a heavier belt is advisable. As an alternative, the conveyor system may be broken up into several flights of any convenient length.

It is not claimed that the two-way conveyor is the answer to all haulage problems in complete belt mining. There will be many cases where it will not even be practical, as in a mining section of very limited length or in an area where tracked supply and mantrip haulage is already installed. However, for mines with large coal reserves, whether in low seam or not, it may be found that two-way belt haulage is applicable, depending on mine layout and extent of development. In many cases where the two-way conveyor can be used, one indicated result will be a reduced unit haulage cost in both directions in addition to a continuous flow of material in both directions without the surges in power and delivery which accompany any system of tracked or rubber-tired haulage.

Summary

As a result of the Weirton study and mine installation, it was found this system accomplishes the following:

1—Rubber-tired equipment for man trips and supplies, cost and maintenance of such equipment, and the necessity for operating personnel are eliminated.

2—Tubular supports for idlers can be used for such purposes as air lines, wires, cable, water, or other carrier lines necessary in modern mines. In many cases this will eliminate the purchase, and installation and maintenance costs, of rib or roof support equipment for these carrier lines. This also applies to pipe for air and water.

3—Time is saved in personnel travel and material handling.

4—There are no empty returns to surface for more men or supplies. Costly driving, grading, supporting, and maintenance of multiple entries formerly needed for these duties are eliminated.

5—Replacement costs are lowered because initial cost of equipment is less.

6—Belt life is increased by reduction of edge wear.

The two-way belt conveyor, therefore, shows promise of improving underground belt transportation because of its multiple uses. However, continued operation under actual mine conditions is necessary to establish the economics on a comparable basis with costs for present-day conventional belt conveyors.

Some Fundamental Principles Applied to the Design And Operation of a Fine Anthracite Plant At Coaldale Colliery

by W. T. Turrall and M. J. Cook

A discussion of modern developments in beneficiation of fine sizes of anthracite, this paper includes a description of the plant flowsheet, an analysis of operating results, and a summary of fundamentals of mineral separation relative to plant operation.

DURING the past several years Lehigh Navigation Coal Co. has installed equipment at Tamaqua and Lansford collieries for cleaning and sizing No. 4 buckwheat −3/32 +3/64 in. and for recovering and cleaning flotation coal −20 +200 mesh at Tamaqua colliery. Papers describing some of these installations have been presented to AIME. At Coaldale colliery, however, methods used to clean fine coal were inadequate and inefficient. Management decided upon a thorough investigation of all existing processes before building a plant to treat the combined three sizes mentioned.

Investigation, carried out on company property and at other plants, entailed a study of hydrotators, hydroclassifiers, tables, spirals, flotation machines, methods of screening, and methods of dewatering. On the basis of this investigation a flowsheet was designed which necessarily excluded many processes in use, and the writers wish to emphasize that no criticism is implied by omission of processes or machines. A determining factor in selection of many machines was the policy of standardizing equipment wherever possible.

Flowsheet and Operational Data

The first principle to be considered in designing of any plant is efficiency of cleaning as related to laws of classification. Beneficiation of coal may be described as separation of two materials having different specific gravities. Although there is an inherent ash content even in the purest coal which slightly affects its specific gravity, it is recognized that many particles are true middlings. The percentage of these particles allowable in the final product depends upon the ultimate ash, or specifications of the consumer.

Richards in his study of free settling and hindered settling classification[1] determined that particles of equal size and different specific gravities had unequal settling velocities. He also determined a definite size relationship for equal settling velocities with free settling and hindered settling conditions. Considering the design of machines used to clean No. 4 and No. 5 buckwheat, it may be assumed that both free settling and hindered settling conditions exist and that ratio of size for practical operations is between the established values. The essential factor is that the more efficiently and closely sized the feed is maintained, the more nearly perfect the resulting separation.

Flowsheet design was governed primarily by factors related to the 1200 to 1500 tons per hr feed to the main plant. It should be stressed that because of the many sources of supply from mine and stripping, the physical characteristics of this feed varied considerably. The most significant influence was varying percentage of fines, necessitating that all machines be capable of operating under maximum peak load during periods of surge. Since cleaning practice is a wet process, about 9000 gal per min containing 14 to 15 pct solids, or 350 tons per hr, must be treated. This slurry contained all the fines, or −3/32 in., but also percentages of oversize, up to 1½ in., dependent on spills or breaks in screen jackets, a not uncommon occurrence in preparation plant practice.

The flowsheet developed is shown in Fig. 1. Circled numbers represent points of sampling. Table I gives operational data including size, ash by size, composite ash, rate of flow in gallons per minute, percent solids dry weight, and average tons of solids per hour. It should be noted that these results are subject to human error in sampling practice. Detailed information reported is a representative evaluation of plant operation.

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Discussion on this paper, TP 3608F, may be sent (2 copies) to AIME before Nov. 30, 1953. Manuscript, March 26, 1953. Fuels Conference ASME and AIME, Philadelphia, October 1952.

Table I. Analyses of Samples Taken from Coaldale New Fine Coal Plant, June 26, 1952

Sample Point on Flowsheet	1* Feed to 24-Ft Hydroclassifier Tank (Calculated)		2 Overflow 24-Ft Hydroclassifier Tank		3 Underflow of 24-Ft Tank, Feed to Oversize Launder Screen (4 Mesh)		4 Oversize from 4-Mesh Screens Returned to Main Breaker		5 Feed to No. 4 Buck Launder Screens (12 Mesh)		6 No. 4 Buck from 12-Mesh Launder Screens, Feed to Wemco Spiral	
Standard Square Mesh	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct
+ 8	1.4	35.76			2.0	35.76	36.8	25.40	0.2		1.5	18.97
10	6.7	28.72	0.4		9.2	29.01	41.9	27.19	6.5	17.80	22.4	25.21
14	7.5	26.02	1.4	13.01	10.0	26.71	10.0	21.95	9.7	19.25	30.2	24.39
20	8.9	24.96	1.4		11.9	25.44	2.8	17.99	11.4	18.50	25.6	23.74
28	11.6	23.97	3.4	15.02	14.8	24.79	1.5	15.67	15.4	20.93	9.4	20.43
35	10.0	23.39	4.0	15.93	12.2	24.39	0.8	13.94	13.4	21.95	2.3	16.06
48	8.1	24.91	4.4	16.53	9.5	26.49	0.7	14.68	9.6	25.24	1.2	14.46
65	6.6	22.11	5.6	15.71	6.9	24.15	0.8	15.20	7.4	24.36	1.1	17.12
100	7.3	21.73	9.6	15.71	6.2	25.31	1.1	18.02	7.0	23.03	1.0	16.15
150	5.2	22.09	8.8	17.63	3.7	25.85	0.2	20.80	4.0	26.23	0.7	19.26
200	4.2	23.76	8.0	20.64	2.6	27.00	0.8	24.61	2.8	26.24	1.1	22.25
−200	22.5	36.12	53.0	36.18	11.0	35.49	2.6	35.21	12.6	34.88	3.5	33.75
Total	100.0		100.0		100.0		100.0		100.0		100.0	
Pct ash, as received	25.12		26.54		24.56		23.48		23.05		21.06	
Rate of flow, gpm	9000		6300		3500		53		3447		673	
Pct solids, dry wt	14.5		7.0		25.6		53.1		25.3		44.0	
Average tons solids per hr	349.8		115.0		250.0		9.0		241.0		95.5	

* Represents tonnage and flow from breaker. Actual flow into 24-ft tank would include an additional 800 gpm in closed circuit carrying 15.2 tons per hr from points 12 and 17.

Sample Point on Flowsheet	7 Feed to No. 5 Buck Launder Screens		8 No. 5 Buck from 24-Mesh Launder Screens, Feed to 16-Ft Hydrotator Classifier		9 Underflow No. 5 Buck Launder Screens, Feed to 15-Ft Tank		10 Under-Flow 15-Ft Tank		11 Overflow of 15-Ft Tank, Feed to Recirculating Pump		12 Overflow of Wemco Spiral to 24-Ft Hydroclassifier	
Standard Mesh	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct
+ 8												
10											0.4	
14	0.2		0.5								2.2	7.49
20	4.2	20.41	6.6	20.85	0.4						3.2	7.63
28	19.3	25.74	24.9	22.59	0.7	25.41	Unable to sample				3.2	8.35
35	17.7	24.56	28.2	23.75	3.7				0.6		2.8	8.32
48	13.2	25.63	14.3	22.89	11.8	24.08			1.6	11.90	3.2	9.20
65	8.5	25.79	8.2	22.34	18.8	22.80			6.4	13.62	4.6	9.80
100	10.2	24.92	4.6	24.98	15.6	24.92			15.0	17.22	8.2	11.91
150	4.7	25.11	2.7	24.54	10.0	25.74			10.0	19.82	7.4	14.63
200	3.5	25.95	2.1	25.44	6.7	26.58			7.6	22.94	6.0	17.49
−200	18.5	35.45	7.9	34.65	32.3	36.52			58.8	35.04	58.8	34.39
Total	100.0		100.0		100.0				100.0		100.0	
Pct ash, as received	25.73		23.58		28.42				27.52		23.98	
Rate of flow, gpm	2774		1240		1534		734		800*		400	
Pct solids, dry wt	19.0		32.2		7.1				6.6		8.5	
Average tons solids per hr	145.5		116.7		28.8		15.1		13.7		8.9	

* Part of this water is used at times on the launder screens for dilution.

Sample Point on Flowsheet	13 Feed to No. 4 Buck, Two 6-Ft Wilmot Hydrotators		14 Refuse from Two 6-Ft Wilmot Hydrotators		15 No. 4 Buck Coal from Two 6-Ft Wilmot Hydrotators		16 Refuse from 16-Ft Wilmot Hydrotator Classifier		17 Overflow 16-Ft Wilmot-Hydrotator Classifier to 24-Ft Hydroclassifier		18 Two 6-In. Spigot Discharge from 16-Ft Hydrotator to Dewatering Screens	
Standard Square Mesh	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct
+ 8	1.7	19.53	4.2	49.49	1.5	7.49	0.1					
10	22.9	23.75	33.1	74.08	21.0	9.60	0.6	68.00				
14	30.7	23.16	35.2	80.92	30.8	11.81	1.8	50.84			0.1	
20	27.5	20.46	19.9	80.49	27.0	13.74	17.9	42.32			4.8	6.75
28	9.6	20.27	5.4	71.50	11.3	15.37	34.4	61.59			22.8	8.65
35	2.9	16.45	0.7	42.68	3.4	15.49	27.4	80.46			27.2	11.52
48	1.3	16.30	0.5	37.27	1.6	15.07	12.0	86.80	0.4		16.1	13.78
65	0.8	18.58	0.3		1.4	16.10	4.0	85.00	1.0	8.66	9.7	17.17
100	0.8	21.72	0.3		0.7	18.95	1.3	68.55	4.4		7.0	21.86
150	0.4	25.03	0.2	42.28	0.3		0.3		7.8	11.05	3.4	25.35
200	0.4	24.44	0.1		0.6	25.74	0.1	48.39	7.8	15.15	2.1	27.87
−200	1.0	32.51	0.1		0.4		0.1		78.6	35.15	6.8	34,12
Total	100.0		100.0		100.0		100.0		100.0		100.0	
Ash, as received	21.99		76.48		11.81		65.36		30.40		13.97	
Rate of flow, gpm									400			
Pct solids, dry wt	61.9		68.7		69.1		67.6		6.1		33.9	
Average tons solids per hr	86.6		13.6		73.0		21.6		6.3		88.8	

Table I is continued on following page

Table I (continued). Analysis of Samples Taken from Coaldale New Fine Coal Plant

Sample Point on Flowsheet	19 Underflow of Dewatering Screens 30 and 35-Mesh		20 No. 5 Buck Coal from Dewatering Shaker		21 Feed to 45-Ft Dorr Hydro-Separator		22 Overflow of 45-Ft Dorr Hydro-Separator to 180-Ft Diam Thickener		23 Underflow of 45-Ft Hydro-Separator, Feed to Two 8x8-Ft Classifying Conditioners		24 Diluted Flotation to Two 6-Cell Banks of Denver No. 30 Flotation Cells	
Standard Square Mesh	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct
+ 8												
10									0.1			
14			Trace		0.2				0.4	14.06		
20	Trace		8.1	6.02	0.8	15.37			1.2			
28	1.8	13.37	32.7	7.99	1.6				2.4	16.70	Same sizing and ash as No. 23 sample.	
35	9.8	10.84	32.3	10.93	5.0	17.91			7.7	19.46		
48	12.9	14.14	14.3	14.17	7.6	20.26	0.2		16.1	21.91		
65	14.4	18.31	6.1	16.42	8.4	20.91	0.6	10.67	18.9	22.66		
100	13.7	23.46	3.6	21.36	11.8	21.60	3.0		22.5	23.69		
150	8.7	27.70	1.5	22.96	10.4	21.90	7.0	11.04	13.1	26.83		
200	6.8	28.15	0.7	25.01	8.0	23.22	9.2	14.34	6.8	33.07		
−200	31.9	36.27	0.7	27.76	46.2	33.75	80.0	33.73	10.8	37.54		
Total	100.0		100.0		100.0		100.0		100.0			
Pct ash, as received	24.48		11.03		26.57		29.19		23.30		23.30	
Rate of flow, gpm	450				7754		7154		600		1120	
Pct solids, dry wt	10.7		66.3		7.3		4.5		34.2		20.0	
Average tons solids per hr	12.8		76.0		145.4		82.8		62.6		62.6	

Sample Point on Flowsheet	25 Refuse from Two 6-Cell Banks of Denver No. 30 Flotation Cells		26 High Ash Coal from No. 6 Cell Re-treated in No. 1 Cell		27 Coal from No. 1 Cell		28 Flotation Coal from Denver Cells Feed to Robins Dewaterizers		29 Underflow of Robins Dewaterizers, Feed to DorrClone		30 Overflow from DorrClone to 45-Ft Hydro-separator	
Standard Square Mesh	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct
+ 8	0.1											
10	0.7	14.01			0.1							
14	1.2				0.2	10.02	0.1					
20	1.9	23.82	0.3		0.6		0.5	7.61	0.2			
28	3.9	36.14	1.2	8.13	1.7	8.10	2.3		1.0	7.63		
35	11.3	62.98	7.5	9.89	8.6	9.45	7.4	9.38	6.0	9.67		
48	18.4	77.03	20.0	13.36	21.1	12.00	16.2	11.08	15.4	10.43		
65	17.9	84.74	24.0	16.10	27.0	13.10	20.8	11.45	20.7	11.48	1.0	
100	20.0	86.50	24.7	19.35	20.9	15.60	25.3	12.92	26.7	13.12	1.0	8.52
150	12.1	88.51	9.8	24.93	9.1	20.13	13.2	15.80	13.0	15.09	3.0	
200	7.6	90.59	4.7	30.03	4.6	25.10	6.5	18.69	7.9	19.34	6.0	7.07
−200	4.9	90.37	7.8	30.56	6.1	30.86	7.7	24.10	9.1	25.76	89.0	26.31
Total	100.0		100.0		100.0		100.0		100.0		100.0	
Pct ash, as received	79.25		18.42		15.23		13.45		13.60		24.26	
Rate of flow, gpm	610						510		410		270	
Pct solids, dry wt	6.0		42.2		33.1		36.1		23.6		3.3	
Average tons solids per hr	10.1						52.5		26.3		2.5	

Sample Point on Flowsheet	31 Flotation Coal from Underflow of DorrClone		32 Flotation Coal Dewatered by Robins Dewaterizers and DorrClones		33 No. 5 Buck and Flotation Coal Blend Loaded Into Railroad Cars		34* Overflow of 180-Ft Thickener Tank	
Standard Square Mesh	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct	Size, Pct	Ash, Pct
+ 8								
10					0.1			
14			0.1		0.5	7.31		
20	0.2		0.4	8.36	5.1	6.35		
28	0.8	7.84	2.0		19.5	8.60		
35	5.8	9.02	7.7	8.86	22.5	11.09		
48	15.7	10.02	17.2	9.84	16.6	12.86		
65	22.4	10.74	21.5	11.31	13.3	8.74		
100	27.2	11.53	25.9	12.09	11.9	13.93		
150	14.2	14.46	13.4	14.49	5.7	16.32		
200	7.0	18.63	6.7	18.04	2.9	19.61		
−200	6.7	27.08	5.1	27.25	1.9	25.87	100.0	66.93
Total	100.0		100.0		100.0		100.0	
Pct ash, as received	12.36		11.93		11.48		66.93	
Rate of flow, gpm	140						7700	
Pct solids, dry wt	52.4		63.9		65.35		0.04	
Average tons solids per hr	23.8		50.0		126.0		0.8	

* There are 5000 gpm of this clarified water recirculated to breaker; balance of clarified water discharges into creek.

Table II is an equipment list giving information as to make and size. It is believed this information used in conjunction with the flowsheet and operational data will prove of value in design or operation of fine coal plants.

Sizing. Hydraulic Classification and Screening

Requirements: It is readily understandable that screening of 9000 gal of slurry would involve considerable capital expenditure for mechanical screening equipment, together with continuing mainte-

Table II. Equipment in Coaldale New Fine Coal Plant

No. 4 and No. 5 Buck Section of Plant	Manufacturer	No.	Size or Capacity	Pump Head, Ft	Motor Rating, HP	Remarks
Hydroclassifier tank		1	24x10 ft			Prefabricated and assembled in field
Feed pump to launder screens	Barrett, Haentjens & Co.	1	3500 gpm	52	75	
No. 4 and No. 5 buck launder screens	Lehigh Navigation Coal Co.	2	3 ft x 8 ft 9 in.			Oversize screen dressed with 4 mesh stainless steel
		3	3x23 ft			No. 4 buck screen dressed with 12 mesh stainless steel
		3	3x23 ft			No. 5 buck screen dressed with 24 mesh stainless steel
No. 4 buck hydrotators	Wilmot Engineering Co.	2	6-ft diam		Two 10	Slate conveyor
			Two 1500 gpm		Two 15	Pump (Barrett, Haentjens & Co.)
					Two 7½	Shaker
			Two 6x19 ft			Screen 6x9 ft of Prizer tread 3/64x1/16x¾ in. 6x9 ft of 3/64 in. round hole
60-in. spiral classifier triple pitch (pitch 4 in. in 12 in.)	Western Machinery Co.	1	5 ft 6 in. x 27 ft 9 in.		10	For dewatering No. 4 buck
No. 5 buck hydrotator classifier	Wilmot Engineering Co.	1	16 ft diam		10	Slate conveyor
			2400 gpm	18	15	Pump (Barrett, Haentjens & Co.)
					5	Agitator
					Two 7½	Shaker
			Two 3x4 ft			Stationary panel screen 30 mesh
			Two 4 ft 4½ in. x 20 ft 1¼ in.			Main dewaterintg screen 35 mesh
Tank classifier		1	15x10 ft			Prefabricated and assembled in field
Recirculating pump	Barrett, Haentjens & Co.	1	1500 gpm	100	50	
Flotation Section of Plant						
Hydroseparator	Dorr Co.	1	45x10 ft		15	Collecting rake speed 1.33 rpm Hydraulic rake lift
Dorrco duplex diaphragm pumps	Dorr Co.	2	90 to 560 gpm		Two 7½	Adjustable stroke
Classifying conditioner	Denver Equipment Co.	2	8 ft diam		Two 10	Conical bottom tank with airlift
Flotation cells, No. 30 Sub-A	Denver Equipment Co.	12	100 cu ft per cell		Six 25	
Double overflow cells					Four 1/3	Froth paddle drive motor
Flotation cell coal pump	Barrett, Haentjens & Co.	1	500 gpm	76	20	
Flotation coal dewaterizers	Hewitt-Robins Inc.	2	4x16 ft		Two 10	4x12-ft section ¼ mm 4x4 ft Section ⅛ mm
Cyclone pump	Barrett, Haentjens & Co.	1	500 gpm	116	30	
DorrClone cyclones	Dorr Co.	2	12 in. diam			
Turbo compressor	Spencer Turbin Co.	1	1350 cfm		7½	Volume at 12-oz pressure
Reagent feeders	Clarkson Co.	8	Model E			
Reagent distributors	Dorr Co.	2			Two ¼	
*Density control	Foxboro Co.	1				For regulating 45-ft hydro. rakes
*Density control	Mine & Smelter Supply Co.	1				For regulating cell density
*Geary-Jennings sampler	Galigher Co.	3			Three ¼	For sampling feed, coal, and refuse

* Installation not completed.

nance cost. Also it was recognized that screening of finer sizes is a difficult problem and that hydraulic classification, which is more practical, should be utilized wherever possible.

Size specifications for fine coal permit a maximum and minimum percentage of oversize and undersize in final products; as a result there is some overlapping of size between coals. The undersize or −20 mesh in No. 4 buckwheat is restricted to approximately 20 pct, whereas No. 5 buckwheat has no lower size limit according to anthracite specifications. In accordance with the laws of classification, efficient cleaning of No. 4 buckwheat hydrotator feed would depend on removing +8 mesh oversize and maintaining −20 mesh as low as possible.

The No. 5 buckwheat at Coaldale is a combined product of coal made by two different processes, each dependent on the other for overall maximum efficiency. Important in this respect is the practical separation of the coarser fraction of the No. 5 buckwheat, i.e. −20 +48 mesh, to be cleaned by hydraulic classification in the 16-ft hydrotator classifier; also important is the classification of a −28 +200 mesh fraction that can be economically cleaned by froth flotation.

Preliminary Classification: The major portion of −35 mesh in the feed of 9000 gal per min flow is removed in a 24x10-ft high hydroclassifier. There are no collecting rakes in this classifier. The sizing is based on settling velocity and rate of flow. The recovered +35 mesh fraction in the underflow is pumped to screens by a 3500 gpm centrifugal pump. Because closed circuits exist with respect to the 24-ft hydroclassifier, i.e. sample points 12 and 17 of Table I (the overflow of the Wemco spiral classifier and the overflow of the 16-ft hydrotator-classifier) the actual flow to the 24-ft hydroclassifier is 9800 gal per min carrying 365 tons per hr of solids. Operating results of this hydroclassifier are shown in Table III. To make a practical separation of the 35 mesh under turbulent conditions an overflow rate or an equivalent settling rate of 140 ft per hr is required.

Table III. Overflow Rate Data for 24-Ft Diam Hydroclassifier

	GPM	Cu Ft Per Hr
Feed*	9800	78,400
Underflow	3500	28,000
Overflow	6300	50,400
Net diameter tank**	21 ft to 3½ in.	
Net area tank	358.7	
Overflow rate, ft per hr†	140.5	

* 9000 gpm breaker feed + 800 gpm recirculated load = 9800 gpm.
** 23 ft 11½ in. − (2 ft x 1 ft 4 in. inside launder) = 21 ft 3½ net diam tank.
† Overflow rate, ft per hr = $\dfrac{\text{Overflow volume, cu ft per hr}}{\text{Area of tank, sq ft}}$

Sample 2 of Table I shows results of preliminary classification. The removal of 115 tons of fines in 6300 gal of overflow has direct bearing on the problem of screen design for sizing No. 4 and No. 5 buckwheat before cleaning.

Launder Screens: Before the remaining 250 tons per hr were screened, the efficiency of several types of screens was investigated. It is well known that with fine sizes, once water is removed screening stops, and only water added by sprays or flooding,

as in pools, will increase the efficiency. Use of both vibrating and reciprocating shakers in other operations verified this condition. Following Parton's study[2] of launder screens in operation at Nesquehoning preparation plant, this type of screen was installed at both the Lansford and Tamaqua plants, having proved more efficient in sizing smaller anthracite coals. Launder screens are stationary, with screen cloth fitted over a wooden grid divided into compartments, or cells, 6x6x6-in. along a launder. In each cell a hole is drilled for an insert bushing or orifice. Since sizing depends on maintenance of suitable hydraulic conditions, essential requirements for operation of launder screens may be listed as follows:

1—Solids in feed should not exceed 25 pct.

2—Slurry should be distributed evenly over the screen.

3—Flow over launder screens should be regulated by orifices of such diameter that essentially all the water will have been removed through the cloth by the time the flow reaches the discharge point.

4—Enough water should be allowed to overflow the end of the screen to provide continuous removal of the product.

5—The slope of the screen, which may or may not be fixed, must maintain sufficient velocity to transport the product to the discharge point.

Table IV contains more specific data relating to launder screens. Operating results of launder screens are revealed by a study of sample points 3 to 9 inclusive, Table I. Fig. 2 is a view of launder screens in operation.

In screening No. 5 buckwheat it is important to note that considerably more fines and a higher dilution report in the product from the screen, see sample point 8 of Table I. This was purposely done to provide the required amount of make-up water for operation of the 16-ft hydrotator classifier.

No. 4 Buckwheat Feed

Dewatering and Classification: It is necessary to provide a dewatering step before feeding to the No. 4 buckwheat cleaning equipment. A 60-in. Wemco spiral classifier, shown in Fig. 3, was installed for this purpose. This method of dewatering has an added advantage of removing by classification the undersize carried with water from launder screens. Sample points 6, 12, and 13 of Table I illustrate performance of this unit. From a feed of 95.5 tons per hr, 8.9 tons of undersize are removed in a flow of 400 gal per min with little loss of +20 mesh. A closed circuit with this overflow has eliminated direct losses.

A change in size characteristics of feed on the second shift made it necessary to install an adjustable overflow weir mechanism, by which the pool area could be decreased from an effective area of 66 sq ft to 44 sq ft. This change in pool area resulted in reduction of the excess −20 mesh by classification and permitted use of the same screens required for day shift operation.

The Wemco spiral has proved an efficient machine during surge periods. Normal feed rate to this unit is approximately 90 tons per hr, but it has successfully handled twice this tonnage with no delay in operating time. It is also more efficient in dewatering and removing undersize than is the conventional tank with drag conveyor used at Lansford and Tamaqua fine coal plants.

Classification of Flotation Feed: Classification results of the 75-ft diam hydroclassifier at Tamaqua, operating under conditions similar to Coaldale requirements, were studied. This classifier, having an equivalent settling rate of 15.9 ft per hr, indicated that classification at 200 mesh was being made with high efficiency.

For the Coaldale operation it was decided to make a separation at 100 mesh. An investigation was conducted to determine the size of hydroseparator to give the most practical classification at this mesh. Fig. 4 shows results of laboratory long tube tests to determine percentage of −100 mesh in the total solids that settled and could be considered underflow product, together with the percentage of +100 mesh in the total solids that were siphoned off and could be considered overflow product.

The curves developed do not indicate any sharp break in classification, and to provide for minimum loss of +100 mesh in the overflow, an arbitrary free settling rate of about 40 ft per hr was selected. At this settling rate 33 pct of −100 mesh reported in the underflow. An attempt was made to confirm this figure by a pilot test using a 12¾-in. diam cone classifier. It was necessary in this test to feed the cone a representative sample from the 9000 gal in the preparation plant discharge flume. Results as shown in Fig. 5 and in Table V represent conditions when the test was run having an equivalent settling rate of 37.3 ft per hr. The +100 mesh loss in the overflow was slightly higher than loss from the long tube test, owing to more turbulent conditions in the cone. The percentage of −100 mesh in the underflow checked reasonably well with the long tube test.

Laboratory tests together with pilot tests indicated that a 45-ft diam hydroseparator should be used, having an inside launder to give a 42-ft 9-in. net diam tank and an equivalent settling rate of 40 ft per hr.

For comparison to the pilot test, actual operating results of the 45-ft hydroseparator having an equivalent settling rate of 40.7 ft per hr are plotted in Fig. 5. Because there was more oversize in the feed in pilot plant tests, results show a slight variation in the coarse sizes. This was to be expected prior to construction of the new plant for No. 4 and No. 5 buckwheat.

Table IV. Launder Screen Data

Launder Screen	Length, Ft	Width, Ft	Total Area	Slope Per Ft	Orifices Per Screen	Size of Orifice, ID	Stainless Steel Screen Cloth Mesh	Wire Diam	Opening In.	Open Area, Pct
Two oversize	8 ft 9 in.	3	52.5	1½ in.	90	1⅜ in.	4	0.047	0.203	65.9
Three No. 4 buck	23 ft ⅞ in.	3	207.6	1⅛ in.	240	13/16 in.*	12	0.023	0.060	51.8
Three No. 5 buck	23 ft ⅞ in.	3	207.6	1½ in.	240	13/16 in.*	24	0.015	0.0267	41.1

* ⅛ in. orifice inserts are used to adjust screen discharge flow to the desired amount for removing solids off screens.

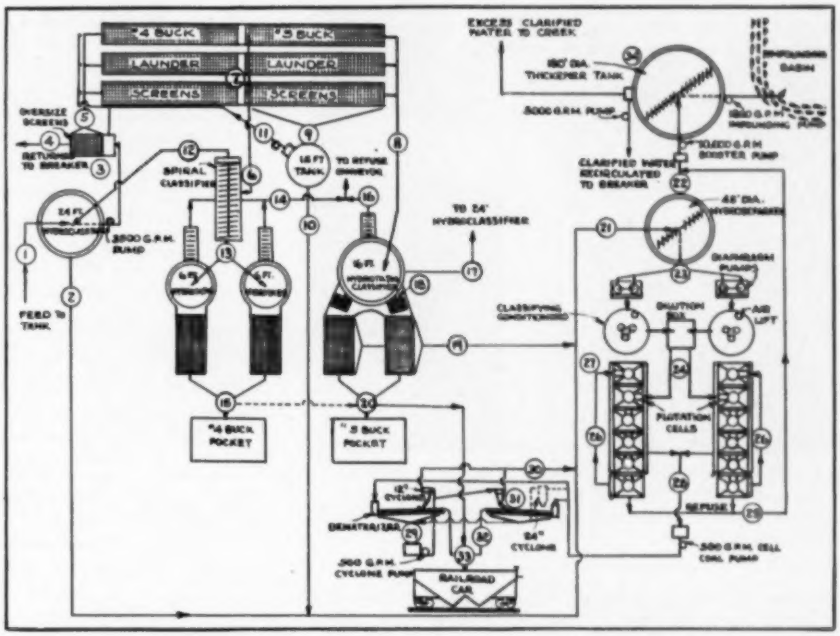

Fig. 1—Flowsheet of the Coaldale new fine coal plant. The circled numbers are sample points.

Theoretically there should be very little −200 mesh in a hydroseparator underflow having this overflow rate, but because of the volume of water required to pump the underflow, which is the same as the overflow water, a normal amount of approximately 10 pct −200 mesh can be expected in feed to flotation cells. The elimination of the greater percentage of −200 mesh in the final coal product is discussed under Dewatering.

Table V. Results of Pilot Classification Test

Item	12¾-In. Test Cone		45-Ft Diam Dorr Hydroseparator	
	GPM	Cu Ft Per Hr	GPM	Cu Ft Per Hr
Feed	4.45	35.6	7754	62,032
Underflow	0.35	2.8	600	4,800
Overflow	4.10	32.8	7154	57,232
Net diam of tank	12¾ in.		42 ft 4 in.*	
Net area of tank	0.88 sq ft		1407.3 sq ft	
Overflow rate, ft per hr**	37.3		40.7	

* 45 ft 0 in. − (2x1 ft 4 in. inside launder) = 42 ft 4 in. net diam of tank.

$$\text{** Overflow rate, ft per hr} = \frac{\text{Volume of overflow, cu ft per hr}}{\text{Area of tank, sq ft}}$$

Sample points 21, 22, and 23 indicate that from a feed of 145.4 tons per hr to the hydroseparator 62.6 tons per hr at 23.3 pct ash is recovered as feed to the flotation plant. The preparation plant waste water carrying the remaining 82.8 tons per hr is pumped to a 180-ft thickener for clarification. Sample point 34 shows that the small loss of solids in the overflow of the 180-ft thickener is all −200 mesh and well within the limitation of 1000 ppm allowed by anti-stream pollution laws. From the overflow of the 180-ft thickener 5000 gal per min are recirculated to the preparation plant. The underflow of this thickener is pumped to a waste impounding basin.

Coal Beneficiation

No. 4 and No. 5 Buckwheat: Beneficiation of No. 4 and No. 5 buckwheat is accomplished in machines employing the same fundamental principle of separation, a separation based on a differential settling rate of particles in a relatively dense media maintained in suspension by an upward rising current.

The design of a Wilmot 6-ft hydrotator, as used in cleaning No. 4 buckwheat, necessitates recovery of fine solids in the water which overflows the separating tank with the cleaned coal. This being the media used for producing the required gravity in the separating tank, added water is eliminated in the recirculating sump overflow. Any build-up of these fines in the circulating load of media has the disadvantage of raising gravity in the separating tank, resulting in a higher ash coal product. For this reason considerable importance was attached to removal of fines by the Wemco Spiral Classifier. Inefficient sizing before cleaning necessitates continuous or intermittent removal of fines from the recirculating system.

Design features of the Wilmot 16-ft hydrotator classifier do not present the same problem. Overflow product of this machine, together with undersize, is removed by dewatering shakers and discharged directly from the circuit.

In the No. 4 machine the pump used for recirculating media and creating the upward rising current is outside the separating tank. In the No. 5

Fig. 2—Launder screens for separating No. 4 and No. 5 buckwheat.

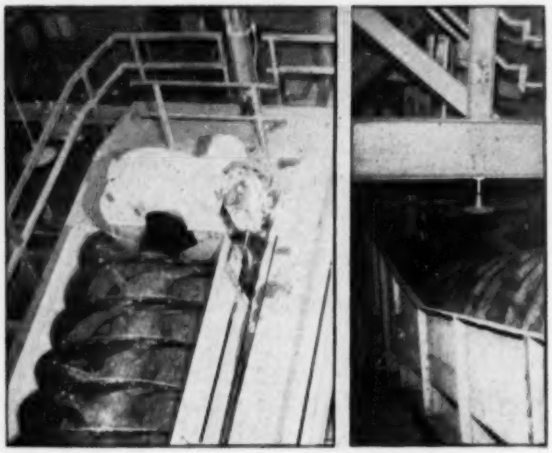

Fig. 3—Two views of the Wemco 60-in. spiral for dewatering and removing undersize in No. 4 buckwheat before it enters cleaning equipment. Pool overflow and dewatered feed are shown.

machine the pump is inside the separating tank and recirculates a specific volume from the top classification zone. There being no closed circuit of fines to the system as in the No. 4 hydrotator, it was considered essential that water and solids required for maintaining the proper gravity classifying condition be obtained from the launder screens.

A comparison of sample points 13 and 8 illustrates feed conditions to the No. 4 and No. 5 machines. Whereas 86.6 tons per hr at 61 pct solids were fed to No. 4 hydrotators, 116.7 tons per hr at 32.2 pct solids were fed to the 16-ft hydrotator classifier. With the increased volume of water required for operation of the 16-ft hydrotator classifier, it is understandable that screening efficiency for No. 5 buckwheat is intentionally lower.

In operation of 6-ft hydrotator and 16-ft hydrotator classifier, effcetive separating gravities in the hindered settling zone are affected by upward rising velocities. Calculated velocities for the No. 4 and No. 5 machines are 427 and 150 ft per hr, respectively. Operating results of the No. 4 hydrotator may be observed by a study of sample points 13, 14, and 15. Fig. 6 and Table VI show float and sink analysis of feed to the No. 4 buckwheat hydrotators, together with calculated cleaning efficiency. An efficiency of 97.4 pct is dependent on the factors previously discussed.

In determining efficiency of the No. 5 buckwheat 16-ft hydrotator classifier, shown in Fig. 7 and Table VII, it is necessary to consider all the products of this machine, i.e. sample points 8, 16, 17, 19, and 20. Normally sample point 16 is considered as cleaner refuse, but since the separating tank overflow, 17, and the screen undersize, 19, are reject products from the hydrotator classifier they have been calculated in the actual refuse on a weight percent basis. Since these products are low in ash, 30.4 and 24.4 pct respectively, they lower the cleaner refuse from 65.36 pct ash to a calculated 47.01 pct. Calculated cleaning efficiency with this lower refuse ash is 78.5 pct. The lower efficiency may not be representative of other plant operations; also it does not represent a loss in the Coaldale flowsheet. Both overflow and screen undersize are fine high ash products, and their rejection permits production of a coarse low ash No. 5 for blending purposes. A study of the flowsheet, Fig. 1, and operating results of the flotation circuit shows the feasibility of this type of circuit, especially when flotation is employed.

Flotation: There are at present two schools of thought pertaining to flotation of anthracite. The processes can be simplified by referring particularly

Table VI. Float-and-Sink Yield Analysis of Feed to No. 4 Buckwheat Hydrotators

Theoretical Recovery of No. 4 Buckwheat from Float-and-Sink Graph, Fig. 6 — Pct 86.5

Actual Recovery Based on Ash Formula

Item	Sample Point No.		Ash, Pct
Feed	13	=	21.99
Coal	15	=	11.81
Refuse	14	=	76.48

$$\text{Recovery} = \frac{\text{Refuse Ash} - \text{Feed Ash}}{\text{Refuse Ash} - \text{Coal Ash}} \times 100 =$$

$$\frac{76.48 - 21.99}{76.48 - 11.81} \times 100 = \quad 84.3$$

$$\text{Cleaning Efficiency} = \frac{\text{Actual Recovery}}{\text{Theoretical Recovery}} \times 100 =$$

$$\frac{84.3}{86.5} \times 100 = \quad 97.4$$

to the physical character of flotation coal on the cell surface, i.e., matte or froth flotation. The processes may be classified as follows:

1—Matte flotation (Lasseter process), mechanical cell, depends on use of greater quantities of flotation reagents to create a bulk or mass suspension of solids on surface. This process employs mechanical agitation to keep solids in suspension but does not use the volume of air required for maintaining a froth.

2—Froth flotation, pneumatic cell, uses a minimum of flotation reagents and employs no mechanical agitation in the cell but requires a large volume of low pressure air to create froth and maintain solids in suspension.

3—Froth flotation, mechanical cell, uses a minimum of flotation reagents and employs mechanical agitation, with or without supercharged air, to create froth and maintain solids in suspension.

Flotation cells now in use which are representative of the above condition are the Denver sub A, Lasseter type; the Steffensen cell; and the Denver sub A. These flotation cells all have respective merits,

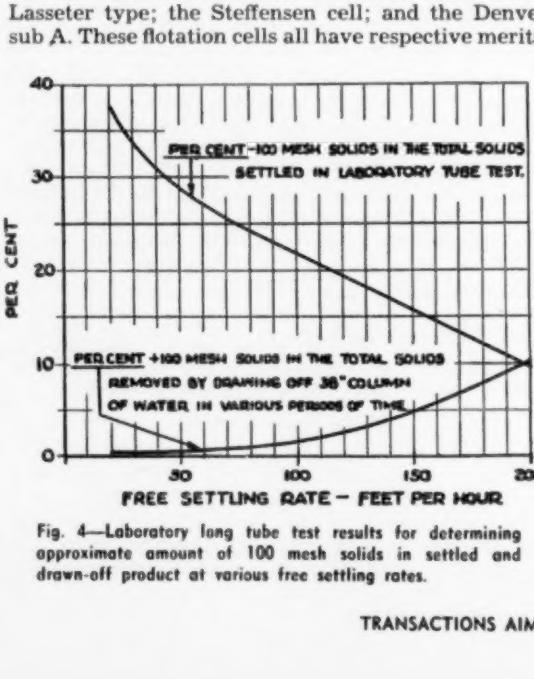

Fig. 4—Laboratory long tube test results for determining approximate amount of 100 mesh solids in settled and drawn-off product at various free settling rates.

particularly the Lasseter cell when it is necessary to float a coarser coal.

Matte flotation vs froth flotation of coal will no doubt be a controversial issue for several years to come. This paper will elaborate on the froth flotation process. Every consideration was given to creating what the writers believed to be fundamental requirements for froth flotation. Unlike metallic mineral flotation, whereby only a small weight percent of the feed is floated, froth flotation requires ample aeration to the pulp to float the mass. Important too are the size and number of bubbles and their dispersion.

Supercharged air at 8-oz. pressure is considered essential. An improvement has been accomplished by use of 16-bladed neoprene impellers instead of

Table VII. Float-and-Sink Yield Analysis of Feed to No. 5 Buckwheat Hydrotator

	Pct
Theoretical Recovery No. 5 Buckwheat from Float-and-Sink Graph, Fig. 7	82.9

Actual Recovery Based on Ash Formula

	Item	Sample Point No.		Ash, Pct
	Feed	8	=	23.58
	Coal	20	=	11.03
Refuse	Cleaner Overflow*	17	=	30.40 × 0.1548
Refuse	Cleaner Refuse	16	=	65.36 × 0.5307
Refuse	Screen Underflow**	19	=	24.48 × 0.3145
				= 47.01 Pct Ash

$$\text{Recovery} = \frac{\text{Refuse Ash} - \text{Feed Ash}}{\text{Refuse Ash} - \text{Coal Ash}} \times 100 =$$

$$\frac{47.01 - 23.58}{47.01 - 11.03} \times 100 = 65.1$$

$$\text{Cleaning Efficiency} = \frac{\text{Actual Recovery}}{\text{Theoretical Recovery}} \times 100 =$$

$$\frac{65.1}{82.9} \times 100 = 78.5$$

* Portion of this is recovered by returning to 24 ft diam hydroclassifier.

** Portion of this is recovered by flotation by directing to 45 ft diam hydroseparator.

the 9-bladed impellers used at Tamaqua. The original test at Tamaqua with two 16-bladed impellers indicated that more lively small bubble froth could be expected, with resulting improved efficiency. Use of supercharged air together with a minimum +28 mesh in feed to flotation permitted lower quantities of reagents, consumption being 0.307 lb per ton feed of pine oil and 0.7 lb per ton feed of No. 2 fuel oil, as compared to Hagen's reported consumption[8] in the Lasseter process of 0.25 lb per ton feed of an amyl ethyl alcohol frother and 5.0 lb per ton feed of kerosene.

Once coal reaches surface in froth flotation it is imperative that quick removal be made to prevent any serious showering of particles back into the pulp. Double overflows and where necessary four paddles instead of two as in the first two cells of each bank made this condition possible. A wetter froth was produced, with approximately 36 pct solids as compared to Tamaqua with single overflow at 46 pct. In this respect the Lasseter process offers a decided advantage in removing a coal product from the cells at about 48 pct solids. Use of double overflows makes effective the total area of cell. This is of fundamental importance in coal flotation. Efficient separation of +28 mesh particles in the feed eliminates any necessity for utilizing matte flotation.

Fig. 8 shows graphically the size consist of a froth flotation feed, together with an ash distribution

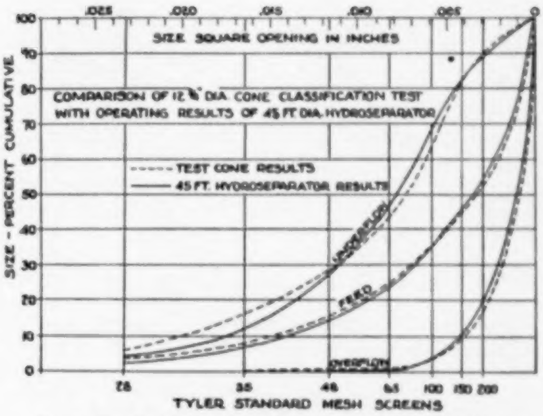

Fig. 5—Results of pilot classification test with 12¾-in. diam cone to determine size of hydroseparator compared with actual operating results of 45-ft diam hydroseparator. Statistics are shown in Table V.

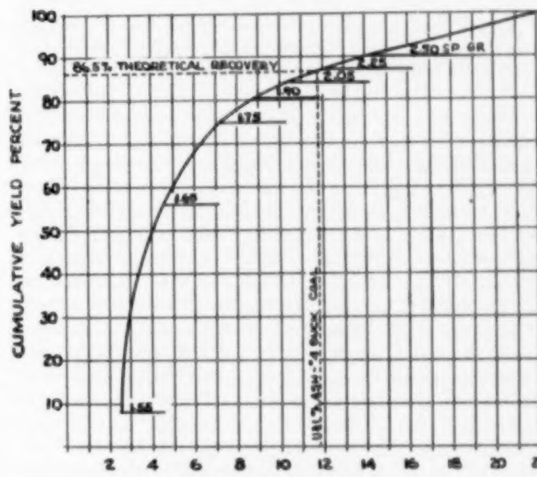

Fig. 6—Float and sink yield data of feed to No. 4 buckwheat hydrotators. Table VI gives accompanying statistics of recovery and cleaning efficiency.

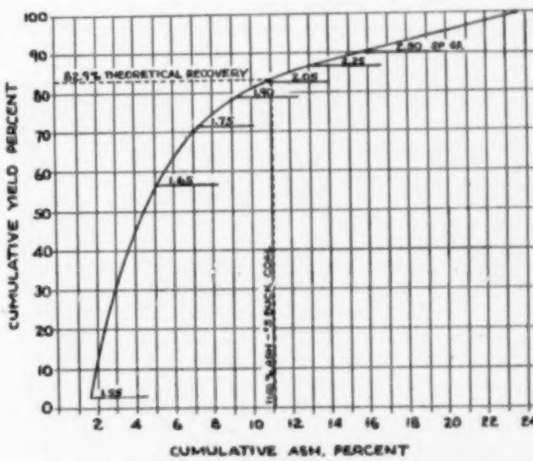

Fig. 7—Float and sink yield data of feed to No. 5 buck equipment. Accompanying statistics of recovery and cleaning efficiency are shown in Table VII.

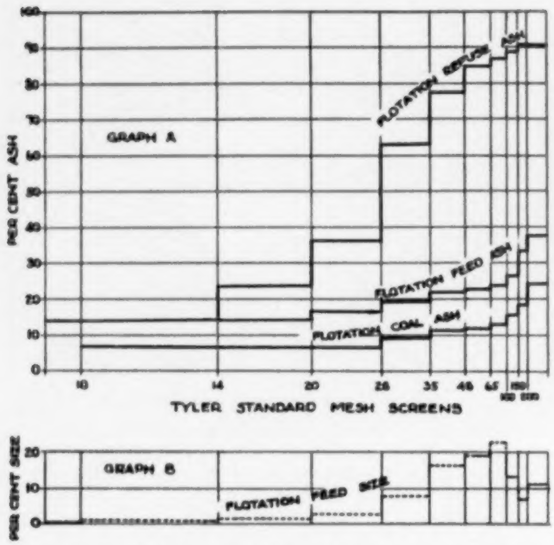

Fig. 8—Graph A, above, indicates ash distribution according to size consist of flotation feed, refuse, and feed. Graph B, below, indicates size consist of flotation feed.

analysis of sizes in feed, refuse, and coal. It is clearly shown that the particle size for efficient froth flotation cleaning should not be larger than 28 mesh.

In a paper describing Tamaqua flotation plant, Parton[4] established that coarse coal required longer floating time and that it was necessary to re-thicken tailings of the primary circuit to maintain the required density of 20 to 25 pct solids to float the coarser coal. This may be better understood when it is realized that about 70 pct of the total coal floated is removed in the first two cells of the primary circuit, resulting in the tailings being reduced to about 10 pct solids.

It is interesting to note that in the Lasseter cell the grids of early Denver sub A cells have been replaced, a development which permits existence of an agitation zone, as well as a quiescent zone where there is less chance of the coarse particles being sheered from the bubbles by forces within the cell. Such a cell design may result in a more uniform flotation rate of all particle sizes.

The study of flotation at Tamaqua revealed that several important changes could be incorporated in the Coaldale flotation circuit to increase efficiency, provided several factors were taken into consideration, namely, elimination of coarse sizes in feed, a uniform feed rate, removal of coarse high ash particles in feed to cells, and the installation of automatic controls.

The flotation circuit treating a minimum of +28 mesh as developed for Coaldale eliminated the necessity of a secondary thickening operation. Two parallel banks of six cells are used. Feed enters cell 2, and cells 2 to 5 inclusive float a finished product. Cell 6 is used as a scavenger and by floating a high ash product it is possible to raise the ash of the tailings. Coal product from cell 6 is returned to cell 1, the cleaner cell. The froth product from cell 1 is a final coal product. Coal not recovered in cell 1 may be recovered, as the tailing is in closed circuit with the other cells.

In coal flotation plants every operator is repeatedly confronted with the variations in ash and size

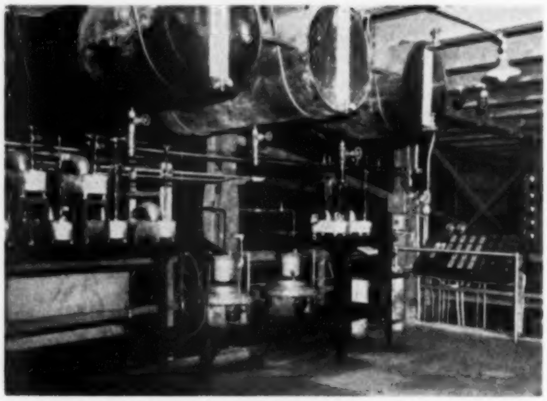

Fig. 9—Reagent feeder, distributor, and equipment control panel.

due to change in character of the feed. Operating conditions at Tamaqua and Coaldale indicated that during such a change feed tonnage may vary ± 50 pct and the ash from 20 to 30 pct.

Design and operation of the 45-ft hydroseparator allow 50-ton storage of feed. This has proved sufficient to take care of surges. Rakes in the hydroseparator have a 24-in. lift and are at present manually controlled by the flotation operator from the control panel. The setting of the rakes is dependent on density of conditioner feed at the discharge of the diaphragm pumps. Raising or lowering the rakes may be automatically controlled when the installation of a Foxborough density control is completed. Changes in rake setting are made when the operator finds a difference in gravity as measured by a pulp density balance. A gravity of 1.20 or 36 pct is maintained for conditioning.

To provide the operator with means of knowing the position of the rakes, company electrical engineers designed a signal system which is simple but invaluable. As the rakes are raised a plow attachment on the hydraulic lift pushes past a series of

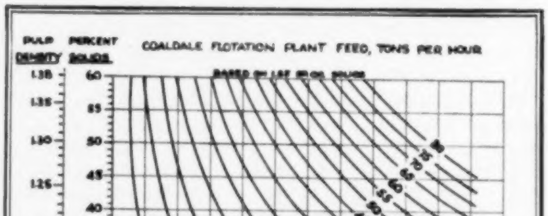

Fig. 10—Capacity of a single Dorr Co. duplex pump in handling flotation feed slurry at various pulp densities and pump strokes.

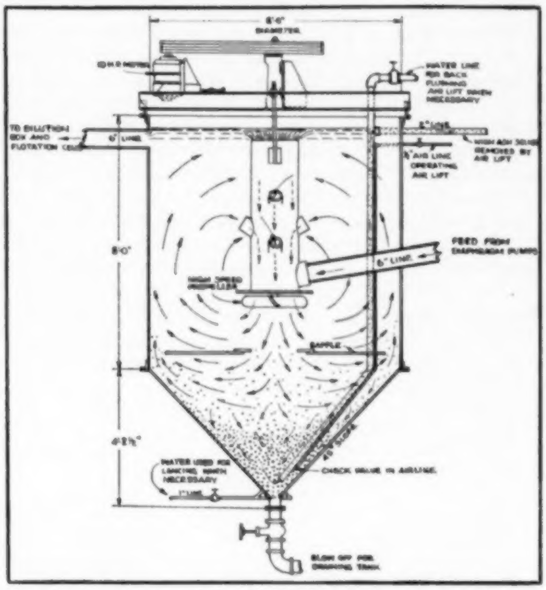

Fig. 11—Details of an 8x8 ft classifying conditioner with airlift.

five sensitive switches equally spaced and each connected to a light on the control panel set at the same spacing. These lights can be seen in Fig. 9, at the right of the photograph. During a period of surge the operator maintains a closer check on feed density and thereby establishes a most important control factor for efficient flotation.

In designing of the plant, provision was made for excess pumping capacity in the diaphragm pumps in case of an emergency. Normally each of these is operated at 300 gal per min rate of flow, but should the signal system indicate the rakes to be approaching the maximum lift, the stroke of the pump is increased until the surge is overcome. Fortunately this requirement is a rare occurrence, as flotation efficiency decreases owing to excessive feed rate.

Fig. 10 charts the capacity of a Dorr Co. duplex pump.

Classifying Conditioner

The classifying conditioner is a development which may prove a contribution to mineral engineering. Its operation at Coaldale colliery has greatly improved efficiency of coal flotation.

At Tamaqua plant it was customary practice regularly to blow off the 12x12-ft conditioner as often as once a shift to remove the accumulated load of pyrite and coarse high ash solids. This material, under severe conditions, accumulated to such extent that it overflowed with the feed to the cells. The Denver sub A flotation cell, although equipped with a sand relief to relieve cells of coarse particles, could not function properly; as a result the cells sanded up, and the froth became dead. With almost no flotation taking place the operator found himself in serious trouble and feed to the plant had to be by-passed to refuse.

A study of cells during such a period indicated that it was essential to maintain a condition in the cells which would permit proper dispersion of air. It was also observed that the accumulation of heavy solids in the cells was giving a short life to impellers, the neoprene rubber being nearly ground off the bottoms, the upper parts of the blades remaining in good shape. A re-designed airlift installed in this conditioner eliminated this condition almost altogether.

Examination of solids that settled in the conditioner in the form of a cone showed that a very high ash fraction accumulated at the bottom. It was decided to investigate the designing of a conditioner that would incorporate a classifying action to separate coarse high ash particles and pyrite from the feed. The resulting machine is shown in Fig. 11. Essentially it is a Denver (Wallace type) super agitator with a classifying zone added. The fundamental principle involves a rising water current as created by the conditioner propeller. Since the propeller is designed to drive the pulp to the bottom of the conditioner, it may be observed that a baffle, having a circular opening in the center and a clearance between the side of the tank, now permits the pulp to drive down into a conical section and then flow in an upward direction through the outer clearance. With such a flow pattern it is necessary only to apply fundamentals of classification to make the necessary separation. To obtain the required rising velocity the hole in the baffle was made 2 ft in diam and so machined that varying size orifice openings could be used. Experiment to date has indicated that with a propeller speed of 240 rpm, an 18-in. orifice opening, and a 6-in. side wall clearance, an efficient separation of the objectionable impurities from the feed is made. A 45° angle cone was selected as a suitable bottom for collecting these impurities. A simple airlift is employed to remove solids sent to waste.

Results shown in Table VIII from an actual test in handling difficult bank material speak for the performance of this new classifying conditioner. In this installation feed to cells discharges through a 6-in. pipe, but in other uses it may be an advantage to overflow the rim of the tank to a collecting launder. To simplify the changing of upward rising velocity a variable speed drive could be utilized. Also in cases where the quantity of solids to be removed from the bottom of the cone is considerable it may be advantageous to use a diaphragm pump to control rate of discharge and percent of solids.

To make the operation of this plant more efficient the Foxborough density control previously mentioned and a Massco-Adams density control for maintaining uniform pulp density to cells are being installed. Galigher automatic sampling equipment

Fig. 12—General view of flotation cell floor.

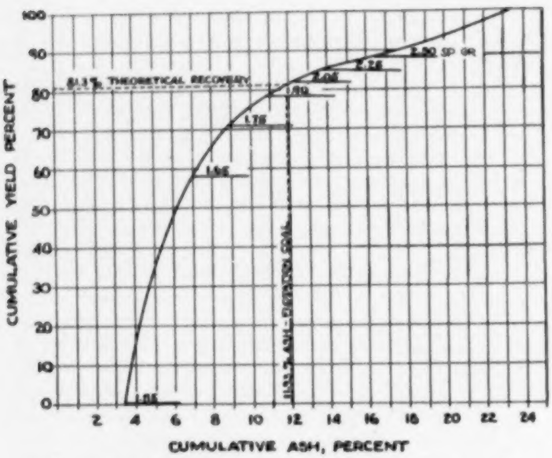

Fig. 13—Float and sink yield data of feed to flotation cells. Analysis of recovery and cleaning efficiency is given in Table X.

will replace hand methods for obtaining samples to represent daily plant performance.

Reagents

Fig. 9 shows the general layout of the reagent feeding equipment. Table IX represents consumption and distribution of reagents in the flotation circuit. Fig. 12 is a general view of the flotation floor with classifying conditioner in background.

Flotation Efficiency

Operating results of the flotation plant are best analyzed by referring to sample points 23 to 28, inclusive. Treating 62.6 tons per hr having an ash

Table VIII. Analyses of Classifying Conditioner Products from an Operation with High Ash Feed

	Feed to Conditioner		Feed to Cells Before Dilution		Airlift Refuse from Conditioner	
Size Mesh	Material, Pct	Ash, Pct	Material, Pct	Ash, Pct	Material, Pct	Ash, Pct
+ 10					0.0	
14	0.0		0.0		0.2	
20	0.3		0.2		0.8	36.28
28	1.5	26.91	1.3	13.16	3.9	56.80
35	12.2	34.08	10.7	20.60	22.6	69.02
48	22.5	37.01	21.4	26.94	30.5	68.47
65	22.6	41.47	22.8	33.04	21.2	73.02
100	23.6	41.96	23.7	39.93	14.2	65.67
150	10.0	45.96	11.3	45.89	4.1	64.96
200	4.5	47.86	5.1	50.49	1.5	63.65
−200	2.8	55.07	3.5	57.96	1.0	62.55
Total	100.0		100.0		100.0	
Composite ash		38.68		33.33		69.39
Solids, pct		33.6		34.4		36.5
Flow, gal per min.		600		540		60
Tons per hr		61.6		54.6		6.5

of 23.3 pct, a refuse of 79.25 pct ash and a coal of 13.45 pct ash is obtained. This is not considered the final flotation product, for in the dewatering system a 12-in. DorrClone, sample point 30, removes 2.5 tons per hr that is 89.0 pct −200 mesh with an ash of 24.26 pct. This is the high ash slime product mechanically entrapped in the froth. Sample point 32 shows the final product having an ash of 11.93 pct. The calculation for cleaning efficiency in Fig. 13 using the above analyses is 98.1 pct, see Table X.

The evidence supporting this flotation circuit to one incorporating a secondary thickening operation is the high ash refuse and the work being done by the scavenger and cleaner cells which have froth products of 18.42 and 15.23 pct ash respectively.

It is not possible in the scope of this paper to discuss fundamentals of dewatering fine anthracite. Papers pertaining to this subject stress the difficulties encountered in this phase of the operation.[5,6]

Primarily surface area is the major factor contributing to efficient dewatering. The more effective the elimination of −200 mesh, the smaller the surface area and the lower the resulting final moisture.

There is no particular problem in dewatering No. 4 and 5 buckwheat, the methods employed utilizing the conventional type of shakers. Sample points 15 and 20 indicate the lowest percentage of moisture from the shakers to be approximately 30 pct. Drainage in the car over a 24-hr period reduces this moisture content considerably.

Table IX. Consumption of Reagents

	Yarmor-F Pine Oil		No. 2 Fuel Oil		Reagent Distribution to Bank of Cells		
	CC Per Min	Pounds* Per Ton of Feed	CC Per Min	Pounds* Per Ton of Feed	Cell No.		Pct
					1	(Recleaner)	0
Two conditioners	44	0.087	200	0.350	2	(Feed)	50
Two 6-cell banks	112	0.220	200	0.350	3		15
Total	156	0.307	400	0.700	4		10
					5		10
Gal per hr	2.47		6.34		6		15

* Based on 62.6 tons of flotation feed, solids per hr.

Dewatering the flotation product presents an entirely different problem. It requires more elaborate equipment and it may include vacuum filtration, centrifugal filters, cyclones and screens. Of these the vacuum filter may be more efficient in lowering moisture content to 20 pct.

The Coaldale plant incorporated screens and cyclones to reduce the moisture content to approximately 33 pct so that it could be blended with the coarse No. 5 buckwheat having a similar moisture. Final drainage of blend product in ground storage or cars permits lowering of moisture to 12 to 14 pct.

The double overflow cells presented an unexpected problem in screen dewatering. It was not anticipated that the percent solids in the froth product would be decreased from 46 pct, as at Tamaqua with single overflow, to 36 pct with double over-

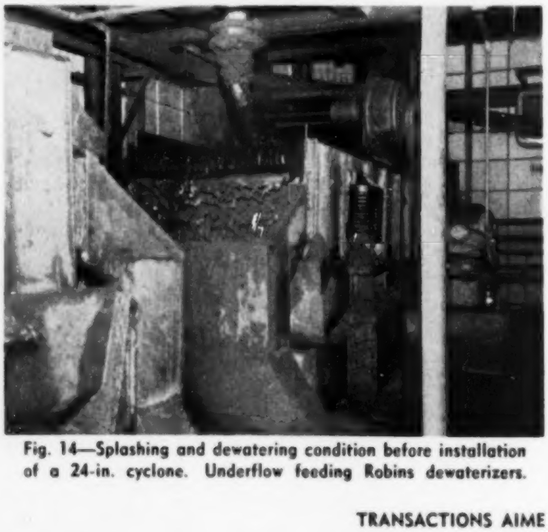

Fig. 14—Splashing and dewatering condition before installation of a 24-in. cyclone. Underflow feeding Robins dewaterizers.

flow. Increase in volume of water pumped on two screens dressed with ¼ mm wedge bar caused a flood condition, illustrated by Fig. 14. It has been necessary to employ the standby unit to dewater to 36 pct moisture.

Use of cyclones in dewatering a flotation product by screens is essential. Samples 28 to 32 show the results by this method. The feed in this circuit is pumped directly to the screen. About 50 pct of the solids having practically the same size consist as the feed passes through the screen openings at 23 pct solids. This is pumped to two 12-in. DorrClones at 28-lb pressure, and the thickened products at 52 pct solids are discharged, for further dewatering, on to the coal layer formed from the feed.

Table X. Float-and-Sink Yield Analysis of Feed to Flotation Cells

	Pct
Theoretical Recovery Flotation from Float-and-Sink graph, Fig. 13	81.3

Actual Recovery Based on Ash Formula

		Sample Point No.	Ash, Pct	
	Feed	23	= 23.30	
	Coal	32	= 11.93	
Refuse	Cell Refuse	25	= 79.25 × 0.8016	68.34 Pct Ash
	Cyclone Effluent	30*	= 24.26 × 0.1984	

$$\text{Recovery} = \frac{\text{Refuse Ash} - \text{Feed Ash}}{\text{Refuse Ash} - \text{Coal Ash}} \times 100 =$$

$$\frac{68.34 - 23.30}{68.34 - 11.93} \times 100 = \frac{45.04}{56.41} \times 100 = 79.8$$

$$\text{Cleaning Efficiency} = \frac{\text{Actual Recovery}}{\text{Theoretical Recovery}} \times 100 =$$

$$\frac{79.8}{81.3} \times 100 = 98.1$$

* Portion of this is recovered by returning to 45-ft hydroseparator.

This circuit has not proved satisfactory for dewatering or plant cleanliness. A circuit now in development stage will overcome the flooding condition as illustrated in Fig. 15 and permit the use of one screen. Flotation product is pumped at low pressure to a 24-in. cyclone having an extra deep cylindrical section. The value of this cylinder has not been definitely established but it is believed that with the high percentage of solids in the feed, classification can be made more efficient. The overflow of the 24-in. cyclone bypasses the screen and is pumped, together with the screen underflow, to one 12-in. cyclone at 28-lb pressure and discharged as in the original circuit.

The value of a cyclone circuit in dewatering flotation coal is in elimination of the −200 mesh. With the secondary cyclone discharging to the 45-ft hydroseparator, any loss of coarse particles is recovered through the closed circuit.

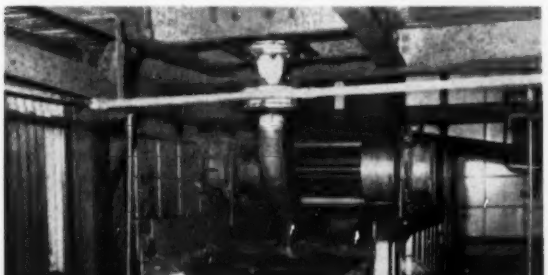

Fig. 15—Improved condition of dewatering with a 24-in. cyclone underflow feeding Robins dewaterizers.

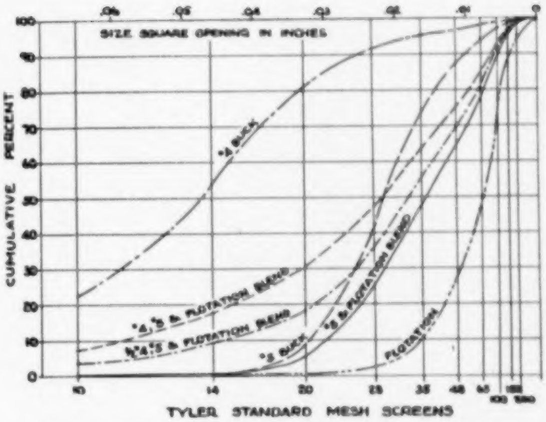

Fig. 16—Size consist of various coals produced from the new fine coal plant.

Considerable study has been made of the effect of moisture resulting from unloading fine coal through hoppers of cars. Experiments in which extra water was added to the dewatered coal have proved that the greater the moisture content, the more compact the product and the more difficult its removal through the hopper.

Conclusion

In summary it may be stated that operation of this plant has more than justified the time required by the mechanical and preparation departments for its design and construction. The ash of the No. 4 and the No. 5 blended product ranges between 11 and 13 pct. Fig. 16 illustrates the size consist of the products which may be made for different market requirements.

Acknowledgments

The authors wish to thank the management for permission to present this paper and also the various members of the operating and engineering staffs of the Lehigh Navigation Coal Co. who designed and constructed this plant. The authors are especially indebted to the operators at Coaldale colliery for their efforts in maintaining high efficiency in the plant and to the staff in the chemical laboratory for the work done on the numerous samples submitted for analysis. Appreciation is also extended to the technical representatives of the many equipment manufacturers who were consulted during the development of the flowsheet.

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The Use of Wooden Rock Bolts in the Day Mines

by Rollin Farmin and Carville E. Sparks

TRIAL installations of rock bolts, of the slit-rod-and-wedge type, were under way at several units of Day Mines, Inc., when Korean hostilities interrupted the already slow deliveries of steel bars to the Coeur d'Alene district. Factory-made bolts had not yet been put on the local market, so the program was halted for lack of supplies. Interest was revived by a visitor's description of wooden roof bolts. These were said to have been used briefly with apparent success in a coal mine, until apprehension voiced by the U. S. Bureau of Mines caused the practice to be suspended.

To make wooden bolts for trial in ground support, Day Mines acquired a second-hand doweling machine equipped with two cutting heads, one to turn out the desired round rods of 2-in. diam, the other to turn out 1-in. rods to be used as powder-tamping sticks. This machine was installed in the all-weather sawmill of the Hercules mine unit at Burke, Idaho, where fabrication of the wooden bolts commenced early in 1951.

Most of the mining in the Coeur d'Alene district is along steeply dipping veins in shaly quartzite and argillite of Algonkian age. Ground support commonly is required in zones where the rocks have been sheared, brecciated, and hydrothermally altered. Pressure from the sidewalls is more troublesome than weight overhead, but both increase with the size of the mine opening. Caving may come from a progressive sloughing of irregular rock fragments or from an exfoliation and buckling of the layered wall rocks. The disintegration is thought to develop from an initial elastic expansion of the rock toward the newly-created mine opening, followed by the dilation of many tiny partings in the rock by absorption of water. As the partings widen, masses of rock develop weight and become free to fall. The function of rock bolts is to prevent or retard widening of partings in the rock supported.

Wooden Bolts, Wedges and Headboards

Bolt assembly used by Day Mines consists of a bolt 4 or 6 ft long, two wedges 16 in. long, and a headboard 30 in. long, Fig. 1. All four pieces are made of local red (Douglas) fir, either green or well-soaked in the mill pond before it enters the sawmill. Bolts are fabricated from cants, 2¼ in. sq, cut from relatively straight-grained timber with a minimum of knots and trimmed to 4- and 6-ft lengths. The bolt then is turned in the doweling machine from 2¼ in. sq to 2 in. diam round, except for a 4-in. length at one end which is left full square to provide the striking head and the shoulder that holds the headboard in position for wedging. The foot end of the bolt is slit with a thin saw for a length of about 16 in., thereby making a slot to receive the wedge against which the bolt is driven for anchorage at the bottom of the rock hole. A similar slit, 12 in. long, is made in the opposite (head) end of the bolt to receive the second wedge, which crowds the headboard against the ground at the collar of the rock hole and puts the bolt in tension. The second slot is aligned 90° from the plane of the first slot to avoid longitudinal splitting and is notched out slightly to allow easier insertion of the collar wedge after the bolt has been driven to bottom. To prevent splitting the headboard by spreading action of the head wedge, this slot is oriented at 90° to the grain of the headboard when the pieces are assembled, Fig. 2. The wedges are similar to standard mine wedges, but more slender; they are cut 1⅞ in. wide and 1 in. thick at the heel and taper out in 16 in. of length.

The headboard, or bearing plate, is not necessary for some types of ground but generally is desirable because it helps the bolt to support an area of loose, friable rock and reduces the tendency for the rock at the collar of the hole to split away from the wedged head by distributing the pressure over a wider rock surface. The headboard may be a 24- to 30-in. length of 3-in. plank, 8 to 12 in. wide, but a similar length of rounded sawmill slab serves equally well at 20 pct of the cost. A hole of 2-in. diam is bored or punched through the center of the headboard, either at 90° or at various high angles to its surface. The bolt is inserted to its shoulder through this hole, then driven into the rock hole.

Bolts, wedges, and headboards are given a full timber preservative treatment to inhibit rot. Bundles of each are immersed in a warm saturated solution of Osmose salts in water for 48 hr, removed, dripped dry, and stored in a relatively humid underground depot to cure.

The current cost of fabricating a wooden bolt assembly follows:

Item	Untreated	Treated
Bolt, 1	$0.13	$0.17
Wedges, 2	0.064	0.08
Slab headboard, 1	0.09	0.18
	$0.284	$0.43

Most wooden rock bolts used by Day Mines are 4 ft long. Holes to receive them, about 42 in. deep and 2⅛ in. in diam, are drilled into the rock to be supported, nearly normal to the periphery of the mine opening. The type of drill used is dictated by convenience: stoper, jackleg, or jumbo-mounted drifter. Correct depth of the hole is assured by use of a measuring stick that has been cut to the proper distance from drill chuck to the ground at the collar of the hole when a standard length drill rod is at the bottom.

The bolt is seated to the shoulder through the hole in the headboard, the foot-end wedge is placed in its slot, and the assembly is inserted into the rock hole. Then the bolt is driven until it is seated solidly on the wedge against the bottom of the rock hole. Driving may be by hand with a sledge, or

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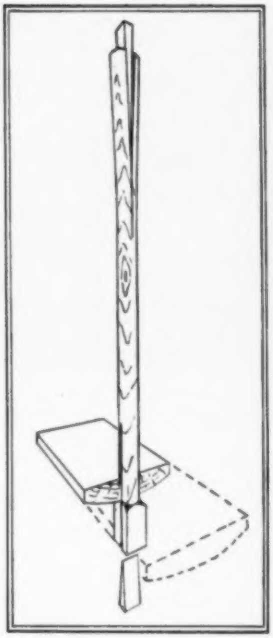

Fig. 1 (Left)—Drawing of a wooden bolt assembly used in the Day mines. Fig. 2 (Right)—Completed installation of bolts and slab headboards in the Dayrock mines. Heads of bolts have been trimmed off, and voids behind headboards have been filled with wedge butts and drill cuttings.

better still with a rock drill plus a rod and driving adapter, Fig. 3. A serviceable adapter may be made by welding a 2-in. length of 3-in. diam pipe to one side of a disc of 1-in. thick steel plate and to the other side welding the front half of a worn-out stoper chuck-bushing. In use, the ring of pipe cups over the head of the bolt and holds the driver in position despite rotation; the bushing on the opposite side of the plate holds the end of the driving rod against the plate to transmit blows from the machine.

The outer wedge is driven into the slot in the head of the bolt with an axe; considerably less force is required here than for seating the bolt on the bottom wedge. After a group of bolts has been installed, the job is completed by sawing off the protruding heads and wedges, close to the headboards, and painting the new tips with a timber preservative. The trimmings are used for filling in voids behind the headboards, together with sand, drill cuttings, or excelsior.

The trial installations of 1951 encompassed many of the types of application where steel bolts have been used: in drifts, crosscuts, stopes, raises, and sumps. These installations were intact a year later except where an inadequate pattern of bolting was used. Timber rot had started only in untreated headboards. The wooden bolt was pronounced a sound device, ready for expanded use. A new level was opened at the bottom of the Dayrock mine early in 1952, and the primary type of ground support has been the wooden bolt and headboard. Fig. 4 illustrates the changed shape of excavation for a crosscut where a conventional timbered opening, 10x10 ft, is converted to a 7x7½-ft bolted arch. The cost of the bolted crosscut is only 53 pct of the cost of the larger timbered crosscut.

Bolt spacing is dictated by the character of the ground and the nature of the opening to be preserved. For example, in the brecciated ground of the Dayrock mine a sequence of seven bolt rings, spaced at 42-in. intervals along the length of the drift, Fig. 5, generally gives adequate support. Elsewhere, the same type of opening may require only a five-bolt ring or merely a single row of bolts along the hanging wall. Assorted patterns are employed in cut-and-fill stopes to restrain overhanging brows of waste rock while the ore is being removed and the fill placed.

Bolts with headboards may be damaged by blasting if they are installed too close to the working face; without headboards they may be placed nearby to reduce overbreak. Thus far, bolts with individual headboards have proved better than long boards pinned with several bolts, at least they are more easily fitted to irregular ground.

In view of the general success of the 2-in. bolt, experiments are being made currently with wooden bolts of smaller diameter. The 1⅞-in. bolt in a 2-in. rock hole appears to be adequate; perhaps

Fig. 3—Driving a wooden rock bolt.

Fig. 4—Looking from the new bolted crosscut toward the former timbered one, which required 47 pct more excavation.

Fig. 5—Soft ground supported by a series of seven-bolt rings. The broken rock in foreground fell during the installation.

slightly smaller ones may serve under proper circumstances.

Recently, the use of wooden roof bolts has been resumed in a coal mine[1] and an investigation by laboratory testing with scale models[2] reports a relatively high performance of wooden bolts compared with steel.

Applicability of Wooden Bolts vs Steel

The experience of Day Mines has led to the following opinions as to the merits of wood vs steel rock bolts.

Economics. The cost of the treated wooden bolt assembly of four pieces is about $0.45 as compared with about $1.50 for similar coverage in steel. Direct labor cost for installing the wooden assembly is about $0.50 as compared with $1.00 to $2.50 for steel under similar circumstances. The investment in a stock of wooden bolts is only one-quarter that required for steel, even less with periodic local fabrication. Special tools are not needed for installing wooden bolts. An impact wrench is needed for the steel bolts, and in some cases different bits, rods, connections, and drills are brought in for the installation; where scattered working places are involved, a considerable investment in extra equipment is required. In very hard rock the larger hole to be drilled for the wooden bolt is a disadvantage favoring the use of steel bolts.

Availability. Wood is always available in ample quantity and is easily shaped with a minimum of shop equipment. Steel, on the contrary, becomes scarce-to-nonavailable during wars or strikes in the steel, coal, and transportation industries.

Flexibility for Installation. The wooden bolt may be shortened readily with a saw to fit the length of a hole from which the collar ground has fallen away. A too-deep hole may be used by first inserting a filler length sawed from a spare bolt. Where the outer half of a wooden bolt has been blasted away by renewed excavation, the bottom half generally remains firmly anchored and may be put back into service when the protruding end is split and re-wedged. Steel bolts, on the other hand, need holes of the proper depth, and once anchored are rendered impotent if a foot or two of collar ground is blasted away. Steel wedges, angle washers, and bearing plates which fall into the ore stream become a mechanical hazard in the crusher plant, especially if the presence of magnetic minerals in the ore makes use of magnetic protection impractical.

Performance. Good anchorage is obtained by wooden bolts in ground too soft for the wedged steel bolt, perhaps because the wood more completely fills the hole and prevents deformation of the soft rock into unfilled hole. The wooden bolt is more or less susceptible to timber rot but is free from attack by corrosive mine water, which may rapidly attack the threads of a steel bolt. Steel is better for building up a composite structure of bolts and channels because of its much greater strength in shear and bending. It also provides a better hanger for pipes and cables, unless the ground is too soft for good anchorage, and may be installed closer to the blasting face without damage than wood.

Safety. Hazards present during installation of wooden bolts are fewer than with steel. The miner uses only the familiar rock drill, whereas with steel he also lifts and operates the heavy impact wrench in an awkward overhead position, standing under ground not yet supported. The wedge which tightens the collar of the wooden bolt is driven from an offset position, after the bolt and headboard have provided partial support for the ground. In tension, 2-in. diam fir is somewhat stronger than 1-in. diam mild steel.[3] Either is competent for the purpose. The relative weakness of wood to shear and bending strains is not usually involved unless wooden bolts are placed in interlocking structures or are subjected to blasting. The hazard of fire among installed wooden bolts does not appear great because of the spacing. On the other hand, steel bolt installations frequently invite electric welding and flame cutting underground.

Acknowledgments

The writers are indebted to Mr. Henry L. Day, President of Day Mines, Inc., for encouragement in the project and for permission to publish data. Many members of the staff and mining crews have contributed to the developments here described.

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Changing Factors in Mine Valuation

by Samuel H. Dolbear

THE value of a mine is basically dependent on its capacity to yield profits. Since the ore must be mined, treated, and sold, some of it in various future years, there is a risk involved as to future costs, selling price, and working conditions. It cannot be expected that the economic condition existing at the time of valuation will continue unchanged for long periods in the future.

During the past 20 years, mineral production in the United States has been conducted under a changing economy in many respects more exacting than that applied to other businesses. There have been increased production incentives, technical aid, exploration of privately owned mineral deposits by government at federal expense, and liberal loans for development and equipment, with risk partially assumed by government. Some of these benefits have been counterbalanced by price ceilings, consumption controls, and stimulation of competition from foreign producers who have been offered the same advantages extended to American operators.

For the present, mines will operate under a government policy directed toward reducing federal aid and control. The tenure of this change will depend upon future elections and the status of foreign relations. War and threat of war are now of the most vital significance to the mineral industries. Other factors which influence cost of production, markets, and price of mine output might be classified as Acts of God or Acts of Government. In some countries expropriation and the difficulty of exporting earnings or investment returns are risks that must be considered by foreign capital. Recognizing that this retards American investment in foreign countries, the Mutual Security Agency offers insurance against such expropriation and guarantees the convertibility of capital and profits.

Since it is impossible to predict with certainty either cost of production or selling prices of metals for long periods, some assumptions must be made as to profits in the future. The basic assumption must be that the price of the company's product will vary in proportion to changes in operating cost. There is often a lag in this reaction, however, for prices of minerals are generally more sensitive to declines and less sensitive to increases than are costs. This reflects in part the resistance of labor to downward wage revision and a corresponding alertness in realizing its share of price advances. Some labor contracts include automatic adjustments to metal prices.

Notwithstanding the complexity of the problems involved and the difficulty of weighing their effect on value, such risks may be appraised with reasonable accuracy and a rate of earnings adopted that is compatible with the risk.

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It is, of course, possible to revert to a yardstick of value such as the *commodity dollar*, which has been advocated from time to time, but while revaluation in 1933 disturbed public confidence, the theoretical gold dollar continues to be the standard of greatest stability. Its gain or loss in purchasing power is reflected ultimately in cost of production and selling price of the mine product. At present 35 dollars are allocated to one ounce of gold.

Measurement of Risk

In the application of the Hoskold and most other formulae, a yearly dividend rate commensurate with the risk involved is set aside out of annual earnings. If the risk is great, this rate may be 15 to 25 pct of the amount invested. The remainder is placed in a sinking fund invested in *safe* securities such as high grade bonds or conservative equities, and the interest or dividends from these securities are added to the sinking fund. The sum of these sinking fund payments and the compounded interest at the end of the mine life is taken as the value of the mine.

Admittedly the decision as to the size of the risk rate is the most difficult element in valuation and one requiring the most exacting consideration. It is necessary to look years ahead in an effort to determine future costs, market prices, demand, competition which may develop, including that of substitutes, and other influences common to the mine and to the region in which it is situated.

Another phase of risk is the enactment of unfavorable legislation, taxes, and what appears to be an alarming spread of nationalization and expropriation. Capital is sometimes *borrowed* from the government to finance strategic production. Such loans may be collectable only out of production and involve no liability otherwise. Valuation in these cases must recognize the effect of such a reduction in liability.

Offsetting some of these risks are the possibilities of mechanization and other cost-reducing discoveries, improvements in mining and treatment methods, new uses for minerals and metals, and normal growth of markets.

In this paper, the terms *risk rate*, *dividend rate*, and *speculative rate* are synonymous. *Safe rate* and *redemption rate* are also used interchangeably. These alternatives are used here because they are commonly found in the literature on mine valuation.

In Michigan, the State Tax Commission has long employed a risk rate of 6 pct in its valuation of iron mines. There the outline of reserves is well established and operating costs and conditions are based on adequate experience.

The following comment on rates appears in the report of the Minnesota Interior Commission on Iron Ore Taxation submitted to the Minnesota Legislature of 1941.[1]

> Most engineers agree that 7 percent for the speculative rate is "an absolute minimum". C. K. Leith in

"Mineral Valuations of the Future" writes as follows:

"In practice a very wide range of interest rates is used, depending on the special local conditions, or upon the practice of particular appraisers. In general, the range has been between 6 and 25 percent, with the trend toward a figure of 8 to 10 percent in recent years for standard mining enterprises in which the factors are well known. Where interest rates as low as 5 or 6 percent are used, there usually have been off-setting corrections or discounts in the other factors in the valuation, such as tonnage, production or costs. In other words, part of the hazard is taken care of there rather than by the general interest rate."

The last two sentences in this quotation indicate the method adopted by the Michigan State Tax Commission in the appraisal of mines in Michigan for the purpose of determining values upon which real estate taxes are to be levied. F. G. Pardee, Mining Engineer and Appraiser of Mines, states that the plan used is to consider separately all the speculative features of the particular mine in direct proportion to their bearing on the valuation and their elimination as far as possible, before the interest rate enters the computations. This method of eliminating the speculative features justifies the use of 6 percent by the Michigan Tax Commission.

The rate commonly used for redemption of capital is 4 percent. It is true that the net interest rate or rate for absolute safety and stability of return at the present time is considerably less than 4 percent. When conditions again become normal the net interest rate will probably assume its former value and the assumed rate of interest should reflect normal conditions inasmuch as it is not selected for the immediate future but for the life of the mining property.

Manufacturing operations commonly employ a 6 pct risk rate. Mineral industries that are essentially manufacturing operations with mining or quarrying as secondary activities may also be justified in using a low figure. Some cement plants are examples of this class.

Under the Hoskold plan of valuation, risk rate is treated as a rate of annual dividends having first claim on profits. After payment of this dividend, the remainder of profits goes into a sinking fund which is, theoretically at least, invested in *safe* securities, formerly high grade bonds yielding 4 pct. The sum of this sinking fund accumulation (together with cumulative interest on the sinking fund) for the life of the operation is the value of the mine. *Present value* calculated by the formula shown under Non-Uniform Annual Income provides for time lapse between the date of estimate and the time when profit is actually realized from operations.

In recent years investment in safe securities yielding 4 pct has been difficult of realization. In 1947 Scudder, Stevens & Clark of New York surveyed funds of endowed colleges in the United States, in the amount of 1.3 billion dollars. The average yield was slightly under 4 pct. Life insurance companies realized somewhat under 3 pct and fire insurance concerns about 3.6 pct during the same period. Interest rates in 1951 to 1952 again established an upward trend. The average realization by endowed colleges in 1952 is believed to be about 4.5 pct. To some extent this is due to the increasing presence of common stocks in conservative portfolios. In valuation of short-lived operations the prevailing interest rates probably should be used. Where the life of the operation is of considerable length it is believed that 4 pct is a conservative figure. The distinction between short and long periods is one best left to judgment at the time of valuation.

Effect of Time on Value

Various segments of an orebody usually vary widely in assay value, and because of width, depth of mining, roof conditions, and other factors which change from year to year, the annual yield may vary widely.

A realistic valuation should be based upon a plan of attack in operations which separates the orebody into segments to be mined during uniform periods of time, usually a year, unless there is likely to be a radical variation in grade and cost at lesser time intervals. Valuation should, therefore, be based upon a plan of operation during which the ore to be mined each year is definitely allocated. The plan of operation must, of course, first determine at what point in the mine the stoping is to be commenced. If the valuation concerns an active operation, then the present location of existing faces is usually, although not always, the governing factor. It is possible, for example, that the value of the enterprise could be enhanced by a change of locus in early operations. The highest value is usually obtained by mining the highest grade ore first. This permits larger additions to sinking fund reserves in the early years of operation, and reduces the burden of interest on invested capital.

As Hoskold so clearly expresses it: "If money could not be employed and a marketable rate of interest obtained for its use, the value of any income or annuity would be equal to that paid at the end of one year, multiplied by the number of years the annuity has to run."

But since money can be loaned or invested to yield an income, the present worth of one dollar, payable at some future time, reflects the loss of such income, and it decreases with the length of the deferment period to the extent that it is almost valueless at the end of 40 years. Ore reserves are seldom valued beyond a 30 to 35-year period. If reserves are large enough to survive 35 years of operation, the residual reserve, i.e., that remaining after the 35 years, is worth, on the basis of Imlay actuarial tables, about 3¢ or less for each estimated profit of one dollar.

Residual Value

But it must be recognized that if the operation is revalued at the end of 35 years, or at some lesser interval, a new capital value is created, since the residual ore, or part of it, is brought within 35 years of realization. Thus ore which for this reason has negligible value today may have an established value one year from today.

It is therefore advisable to provide a statement of ore reserves expected to remain after 35 years, or at the end of the terminal date adopted, and the possibility of additional earnings in a period of post-terminal operations.

Uniform Annual Income

Occasionally mineral prices and costs of operation are of such a character that there is a substantial uniformity of income over a relatively short period, possibly two or three years. Many valuations have applied this supposition to longer periods by taking as bases what are considered averages of annual production value and costs. This is the basic assumption in the Hoskold premise. Such a procedure is believed to be faulty, for the presence of so many variables eliminates any possibility of uniformity over long periods. As pointed out elsewhere, higher grade parts of the orebody yielding maximum profits

in the early history of operation may produce a substantial increase in value as compared with the treatment of ore of an average grade during that period. Valuation based on the use of averages over the life of the mine is seldom if ever justified. The practice of maintaining a more or less uniform grade of ore in mill feed may at times produce some tax advantages, but it may be expected to yield a lower valuation.

Non-Uniform Annual Income

The use of formulae based on non-uniform income should be founded upon a careful plan of future production and an exhaustive weighing of all the factors that may influence financial results on a year-by-year basis for the life of the operation. This is especially important in a time such as the present, when results are dependent to a considerable degree on war or threat of war, tax legislation, and government controls and regulations, as well as prices and cost changes.

Often it may be desirable to develop two or more plans of operation, making spot checks on a specific year's results under each of the plans to determine procedure producing the optimum result. Calculation of profits should be made for each year separately, taking into account all predictable factors such as grade of ore to be mined, width of orebody, and roof conditions. Unpredictable costs, as stated, should be absorbed in the risk rate.

The Parks formula, developed to overcome this objection, has been discussed in detail elsewhere.[2] The formula applied to each year of unequal profit is as follows:

$$Vp = \frac{(\text{sum of } PmR^{n-m} \text{ series})}{1 + r' \dfrac{R^n - 1}{r}}$$

in which

Vp = present value as of the date of the valuation
P = yearly profit (may vary each year)
R = \$1.00 + 1 year's interest at r rate
n = life of operation, in years
m = specific year in which income P is received
r' = speculative (risk) rate
r = safe rate

Depletion

The term *depletion* has been defined by E. S. Benson[3] as follows:

> Depletion from the tax viewpoint means the statutory deduction from gross income designed to permit the return to the taxpayer of the capital consumed in the production and sale of a natural resource. The mining enterprise realizes income on the extraction and sale of minerals and a portion of the income realized represents capital consumed. As the resource is exhausted, the mining enterprise approaches the end of its existence unless new sources of supply can be acquired. Depletion from the tax viewpoint is a creature of statute with limited meaning and application and, in essence, is a method for amortizing the value of the primary asset of a mining enterprise.

Determination of the amount of depletion allowable has become a complex matter, with changes taking place from time to time as new legislation is enacted and new rules are formulated by the Treasury Department. The present laws permit depletion on about 60 minerals and metals at rates ranging from 5 to 23 pct, to the extent that depletion may not exceed 50 pct of net earnings.

In the calculation of net earnings and their reduction to *present worth* by the Hoskold or Parks formulae, deduction is made for estimated income taxes, and such deduction should not, of course, be applied to that portion of income arising from depletion. Depletion accumulations each year should be added to "net profit after taxes" in so far as valuation procedure is concerned.

Depreciation

The rate of depreciation used in financial records of mines is usually the maximum rate permitted under income tax regulations of the Bureau of Internal Revenue. These rates are based on the Bureau tables of average useful life ranging in various structures and equipment from 2 to 30 years. If the useful life is actually below these average figures, then the materials concerned may be non-existent before they are fully amortized.

In other cases material may be fully depreciated on the books, but the property may still have substantial remaining usefulness and therefore value. Under wartime certificates of necessity, plants have been amortized in periods as short as 5 years. This is permitted on the assumption that the plant would have little if any value except under emergency conditions. It is possible, however, that substantial value may remain at the end of so brief a period. The actual value, therefore, can be determined only by appraisal. Book value is an academic figure which might be far different from appraised value.

Markets and Competition

Many minerals have market characteristics that must be weighted in determination of risk. Coal is subject to special seasonal factors such as weather, to the control of output by labor in an extraordinary degree, and to the competition of other fuels. Iron and alloy metals are so closely related to military requirements that it is difficult to determine a normal market. The outlook for fertilizer minerals is based upon agricultural demand and is highly sensitive to farm income. Sales of fertilizer decrease promptly with a drop in farm income notwithstanding intensive educational work, which has offset some of that tendency.

Gold mining, with government limitation of price and no freedom of market; restriction in the use of supplies in emergency periods; and deterioration of mine and equipment in periods of idleness presents special problems in forecasting future value. Materials that are dependent on certain markets may suffer at times. When restrictions are imposed on building construction, for example, a long list of minerals and metals may be affected. On the other hand, war demand sometimes creates temporary prosperity for minerals not normally in demand at profitable levels.

Quicksilver, manganese, and chromite are examples of these in the United States, although generally speaking, in none of these are reserves found developed sufficiently for accurate valuation. And there are many marginal deposits of other materials, including staples such as base metals, that cannot operate on anything short of an emergency or war economy. Demand for building materials, cement, gypsum, sand, gravel, and others, is related to general prosperity and to cumulative demand for housing and engineering construction. Demand is reduced when scarce materials such as are used in electrical and plumbing fixtures are diverted into military channels. The influence of new demand based on inventions and engineering developments

is particularly important in this era of atomic progress and jet propulsion, when there is need for high-temperature-resistant materials. The prospect of large production of titanium metal is one example of demand based on technical progress. Expansion of production often means reduction of price, or perhaps this should be stated conversely, since the reduction of price may also be responsible for larger demand. The price history of aluminum is a case in point.

Competition, actual or potential, is an extremely important risk factor, and strangely, one that is even now too often ignored in expansion of mineral production. Uncontrolled increase in the number of potash mines in Germany, years ago, led to bankruptcy of dozens of producers and a reorganization of the industry under government control. The prosperous potash production in the United States could conceivably suffer a similar fate by overexpansion.

Captive mines usually enjoy more security of markets than independent operations. Coal mines operated as railway subsidiaries, iron mines controlled by steel mills, and asbestos production going into the manufacturing operations of a parent company are examples of this sort.

The threat of the development of substitutes is always present. Unless there is specific evidence of danger from this source it can usually be disregarded, for expanding markets are usually sufficient to offset loss by substitution. The widespread use of plastics in place of metal has apparently had little visible effect on demand. Just what the present position of metals would be in the absence of plastics might of course be considered. But, as stated, specific evidence of possible substitutes should be investigated, as for example, the proposal to replace much of the antimony in storage batteries with calcium.

The development of substitutes and their effective replacement of other materials requires considerable time, often many years. The change is usually gradual enough to permit absorption of the competition over substantial periods and need not therefore have any abrupt effect on value.

On a long range basis it is important to study the growth or decline in demand by examination of statistics over a period of years. The price trend should be treated in similar fashion. Graphs illustrating the history of production, consumption, and prices are extremely useful, and the reason for any anomalies in the pattern should be sought. Whatever the reason, statistics should be interpreted in terms of risk.

Valuation of Mining Shares

Procedure for the valuation of corporate shares differs from that used for a mining property. Under present tax regulations depletion allowance to a shareholder is limited to the purchase price paid for the stock. Presumably this is restricted to the amount paid by the original purchaser, and it is doubtful if a case can be made for a transferee. It cannot be distributed to the shareholder until earnings have first been fully paid out. Since tax legislation and regulations change from year to year, valuation must be based upon the legal situation at the time of valuation. The specific tax background is usually different for each corporation, and this makes an expert detailed analysis essential.

The value of shares is also affected by the dividend policy of the company. In most cases current earnings are not completely distributed as dividends. That part of earnings carried into reserves comprises a residual value and its disposition may have a profound effect on the value of shares.

Dividend payments may not correspond to the risk rate used in mine valuation, and earnings carried into reserves may be more or less than the sinking fund payments in the mine valuation procedure. The value of shares is therefore dependent in a considerable degree upon the financial policy of the company, at least in so far as a minority shareholder is concerned. If the valuation is made on behalf of the holders of shares enough to control the company's disposition of earnings, then it may be desirable to make one valuation based on the current policy of the company, and another dependent on an adoption of changes in procedure that would yield the optimum results.

Periodic Revaluation

Revaluation made annually or at some other regular interval is in effect a periodic re-audit of the earning power and value of ore reserves. Annual audit of the financial position of an operation is of course generally accepted practice.

The ore deposit, although it is the most valuable asset of the company and may undergo radical changes in value from time to time, is nevertheless usually placed on the books at some pre-determined figure, and except for deductions for depletion, the original value remains unchanged.

Price increases may transform marginal into profitable ores and price decreases may erase parts of the orebody. Under these circumstances the estimated life of the mine and its value undergoes change. For example, when the price of gold was increased from $20.65 to $35 an ounce, gold mines were revalued to establish the profit position and property value at the new figure. Mines with by-product gold were also affected.

Failure to revalue mines may bring about conditions present in two large operations revalued by the writer. The book value of one mine had been reduced to about $750,000 although earnings exceeded $2,000,000 annually and ore reserves for many years' operations had been developed. The application of *discovery value* raised the book value to about 20 million dollars.

To cite another instance, the mine of an American-controlled foreign corporation had been valued by income tax authorities at $750,000 in 1919. Since this was a highly profitable operation, depletion soon eliminated this figure, and when the mine was sold 20 years later for more than $8,000,000 it was discovered that over-depletion amounted to around two million dollars. Tax authorities took the position that two million from over-depletion, plus the sales price, a total of $10,000,000, was not a repayment of capital. They classified it as net earnings for the year in which the sale took place.

Periodic revaluation would of course have avoided the penalty for this oversight. Furthermore it provides information of importance to management and is of concern to shareholders.

References

[1] Minnesota Interior Commission on Iron Ore Taxation: Report to Minnesota Legislature (1941) p. 61.

[2] Samuel H. Dolbear: Periodic Mine Revaluation. *Mining Congress Journal*, December 1952.

[3] K. S. Benson: "Depletion" in Federal Income Taxation of Mines. MINING ENGINEERING (July 1951) pp. 612-616.

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aime NEWS

San Elizario Mission is one of three at El Paso dating from Pope's Pueblo Indian uprising during the 1680's.

Record Gathering Expected for Regional Meeting

From preliminary reports it can be gathered that the Regional Meeting of the AIME scheduled for El Paso October 28 to Nov. 2 will contribute to what promises to be a record crowd at the 1953 version of International Mining Days.

Mining men from all over the U.S., Canada, and Mexico are expected to attend the gathering. The New Mexico Mining Association is holding its annual meeting during the event.

Ben Roberts, chairman of the Mining Committee of the El Paso Chamber of Commerce, co-sponsors of the meet, reports that plans are not complete, but that several speakers have been secured.

"We have been most fortunate in securing Felix Wormser, Asst. Secretary of the Interior for Mineral Resources, to speak at our Welcoming Luncheon," he said, "and also, a man well-versed in the problems of the mining industry, Congressman Ken Regan of the 16th Texas Congressional District, will speak at the general luncheon and at the main meeting of the New Mexico Mining Association."

All International Mining Days '53 subcommittees are hard at work completing final plans for the convention, and all arrangements will be completed by late September. A complete schedule of events will be published in the October issue of MINING ENGINEERING.

Chairmen of the various subcommittees are:

Entertainment—Homer C. Hirsch, Hirsch Bros. Machinery Co., El Paso; Executive Committee—Ben D. Roberts, El Paso Smelting Works, El Paso; Hotel & Housing Committee—Ted Lind, Westinghouse Electric Company, El Paso; Invitations Committee—Mallory L. Miller, Colorado Fuel & Iron Co., El Paso; Publicity Committee—Hawley Richeson, El Paso Chamber of Commerce; Ranchero Breakfast Committee—A. L. Washburn, Don A. Carpenter Co., El Paso; Registration Committee—E. W. McQuade, The Mine & Smelter Supply Co., El Paso; Suppliers Party Committee—Gus Momsen, Momsen-Dunnegan-Ryan Co., El Paso; Technical Programs Committee—E. M. Thomas, Dean of Engineering, Texas Western College.

Mr. Roberts also announced the appointment of Milton E. Hopper of the El Paso Chamber of Commerce as Secretary of International Mining Days '53, succeeding George T. Cates, who resigned. Hopper was formerly connected with the U. S. Department of Commerce.

Hotel reservation requests should be sent to Mr. Ted Lind, Chairman Hotel & Housing Committee, c/o El Paso Chamber of Commerce, 310 San Francisco St., El Paso, Texas. The Paso Del Norte Hotel is headquarters for the convention.

(Program on page 932)

Annual Joint Fuels Session at Chicago

The Annual Joint Fuels Conference, sponsored by the Coal Div. of the AIME and Fuels Div. of the ASME will be held at the Conrad Hilton Hotel, Chicago, October 29 and 30.

Chicago Section of the AIME and the ASME are cooperating in the gathering. Carl T. Hayden is General Chairman, while John R. Michel is Co-chairman of the conference. No special program has yet been formulated for the ladies. However, they are cordially invited by the committee to attend the conference.

Registration fee has been fixed at $3.00. Luncheon tickets are $3.50 each and the charge per person for the banquet is $6.50. Price for the two luncheons on the schedule and for the banquet is $12.50.

General Committee members are: R. H. Bacon, Oliver Campbell, A. Cowie, Nelson L. Davis, C. E. De Leuw, D. S. Frank, John Garcia, E. J. Gardner, T. L. Garwood, P. Hoban, J. S. Kozacka, E. E. Lundgren, P. R. Nichols, F. H. Reed, Glenn Shaeffer, R. L. Sutherland, and R. H. Swallow.

(Program on page 933)

Committees Begin Work on 1954 General Meeting

Committees are busily at work planning programs, papers, and other facets of the 1954 General Meeting scheduled February 15 to 18, New York City. Sessions will be held at the Statler and McAlpin Hotels. Mining Branch headquarters will be at the Statler.

Committees are already at work arranging technical sessions, special events, and social activities. Reporting for the Program Committee of the Geology Subdivision, Chairman F. S. Turneaure states that excellent progress has been made. Projected plans call for papers on origin of various types of iron ores, structural features of the porphyry copper deposits, and lead-zinc and silver deposits. One general session is to be arranged featuring papers on a variety of topics.

Minerals Beneficiation

Minerals Beneficiation will hold its executive meeting Sunday, Feb. 14. A business meeting takes place the following day, prior to a joint meeting with Minerals Industry Education Div. on *Training for Supervisors and Top Management.* A materials handling session is also on the tentative schedule. Plans for MBD, as in the case of all other groups, are by no means finalized. The Scotch Breakfast is slated for Tuesday morning. Other sessions on tap include: crushing and grinding; operating control; concentration fundamentals; joint meeting with Extractive Metallurgy Div. on hydrometallurgical processes; concentration plant practice; solid-fluid separation; pyrolysis and agglomeration; and a symposium, *What's New in Milling.*

MBD will hold its usual luncheon and cleanup technical sessions on the last day of the Annual Meeting.

The Coal Div. expects to hold several joint sessions with the Canadian Institute of Mining and Metallurgy. Proposed first meeting of the groups will cover *Optimum Recovery by Mechanized and Continuous Mining Methods*, by L. E. Young, and *Long Wall Mechanization,* by Frank Doxey.

Among other papers to be presented before the Coal Div. are: *Recovery Circuits for Dense Media Processes*, John Griffen; *New Dry Method for Cleaning Fine Coal*, J. Visman; *New Wet Method for Cleaning Coal*, G. A. Vissac; *Wetting Agents in the Flotation of Coal*, Shiou-Chuan Sun and Delaskar Giraldo-Galvez; *Induced Caving of Anthracite Beds*, Andrew Allan, Jr. and Russell S. Davies; *Productivity on Pitch*, H. Wilton-Clark; *Mechanization on Pitch*, Harold McMillan; *Ventilation and Dust Control for Continuous Mining*, J. E. Elkin; *Remote Control for Continuous Mining*, A. L. Lee; *The Relation of Roof Control to Cross Sectional Shapes Made by Continuous Mining Machines*, Louis A. Panek; *Symposium on Control of Gob Pile Fires*, H. F. Hebley; *Non-Fuel Uses of Anthracite Coal*, Raymond C. Johnson; and *Non-Fuel Uses of Bituminous Coal*, Harold J. Rose. A strip mining session is also contemplated but as yet authors have not been committed to topics.

Other papers at coal sessions will be *Developments of a Tentative Laboratory Testing Procedure for Commercial Bituminous Single Rotary Stokers*, by R. J. Grace; and *Underground Gasification*, by James D. Forrester.

Geophysics

Geophysics Subdivision planning has reached the stage where papers are being actively compiled. A joint session is in the making with the Geology Subdivision. Two symposia are fairly definite: *Depth Determination in Electrical Prospecting*, and *Geochemistry*. Deep bore hole surveying and aeromagnetics have also been suggested as topics.

One session has been definitely organized thus far for the Mining Subdivision. Lamar Weaver has been the guiding force behind operations of this group. *White Pine Mining Operations* will be presented by R. F. Moe; *Storke Level Developments at Climax Molybdenum*, by E. J. Eisenach and Ed Matsen; *Concrete Transportation by Pipe Underground at Cleveland-Cliffs, Ishpeming, Mich.*, by H. C. Swanson; and *Ivanhoe Shaft Sinking Operations of New Jersey Zinc Co. at Austinville, Va.*

Mineral Economics

Charles W. Merrill and Granville S. Borden have been working on a program for the Mineral Economics Div. It has been decided to build the program around these eight general topics: tenancy, taxation, subsidies, international trade and tariffs, finance, monetary stability and gold, emergency controls, and labor policy.

One of the outstanding social events of the 1954 Annual Meeting will be the Mining Branch Dinner. Held for the first time at Los Angeles last year, the dinner met with quick approval of those attending. Speaker for the 1954 edition will be D. F. Kidd.

Program
El Paso Regional Meeting
MINING GEOLOGY

THURSDAY, OCTOBER 29

Chairmen: Lloyd Nelson and Howard Quinn

1:30 to 5 pm

Thomas Clendenin: *Geologic Notes on the Santa Eulalia District, Chihuahua*
William Hewitt: *Geologic Notes on the Santa Eulalia District, Chihuahua*
William Hewitt: *Coahuila Fluorspar.*

INDUSTRIAL MINERALS

FRIDAY, OCTOBER 30

Chairmen: John Allen and Edwin McKee

8:30 am to 12 m

Leonard C. Halpenny: *Water Resources of Arizona*
Clyde S. Conover: *Water Situation in New Mexico with Primary Emphasis on Ground Water*
George A. White: *Resume of Potash Development in the Carlsbad Area* July 1952 to July 1593
A. R. Shride: *Localization of Arizona Asbestos Deposits*
Robert Balk: *Spurrite in the Tres Hermanas Mts., N. M.*

12 m Joint Mining Geology and Industrial Minerals Luncheon

SATURDAY, OCTOBER 31

Chairmen: J. P. Smith and John Wood

8:30 am to 12 m

John Allen: *Bentonitic Shales of the Upper Mesa Verde, San Juan Basin, N. M.*
George Kiersch: *Industrial Minerals Survey of the Navajo-Hopi Reservations, Ariz.*
John W. Anthony: *Gypsum Deposits of the Navajo Reservation, Ariz.*
Eugene Callahan: *Review of the Perlite Industry*
Paul Howell: *Bleaching Clays of Tertiary Age*
Wesley Peirce: *Building Stones*
Robert Wilson: *Bentonitic Clays of the Chinle*

Chairman: Eugene Callahan

1:30 to 3:30 pm

Henry A. Hulsill: *All Absorbents from Diatomaceous Earth in New Mexico*
Note: Two additional papers expected.

FIELD TRIPS

Sunday, November 1: Trip to Carlsbad Caverns under the leadership of Joe P. Smith, Chairman, and assistants.

Monday, November 2: Trip through U. S. Potash and/or other mines under leadership of Joe P. Smith and assistants.

Program
Joint AIME-ASME Fuels Conference

Grand Ballroom, Conrad Hilton Hotel, Chicago

THURSDAY, OCTOBER 29

Chairmen: R. E. Zimmerman, U. S. Steel Corp., Pittsburgh, and Orville R. Lyons, Republic Steel Corp.

10:00 am
Occurrence and Determination of Germanium in Coal Ash from Power Plants: R. C. Corey, J. W. Myers, W. A. Selvig, and L. B. Berger, U. S. Bureau of Mines, Pittsburgh.

11:00 am
Economical Reduction of Moisture Content of Washed Coal, Charles E. Silverblatt and Donald A. Dahlstrom, Northwestern University, Evanston, Ill.

12:30 pm

Luncheon Meeting

Coal and World Affairs: Tom Pickett, executive vice-president, National Coal Assn., Washington, D. C.

Afternoon Meeting

Chairmen: C. H. Sawyer, Eastern Gas & Fuel Associates, Pittsburgh, and Howard A. Herder, Sahara Coal Co., Chicago

2:00 pm
New Developments in the Heat Drying of Coal: W. W. Coffin, manager of dryer div., Link-Belt Co., Chicago.

3:00 pm
Frontiers in Heat Extraction from the Combustion Gases of Coal: E. R. Kaiser, assistant director of research, Bituminous Coal Research Inc., Columbus.

6:00 pm

Cocktail Hour

7:00 pm

Banquet

Toastmaster: Alex D. Bailey
Welcome: F. S. Blackall, Jr., President, ASME
Andrew Fletcher, President, AIME
Speaker: H. W. Johnson, vice president, Inland Steel Co., Chicago.
Subject: History of Steel Industry in the Chicago Area

Presentation of Percy Nicholls Award for 1953 to:
HENRY F. HEBLEY
Pittsburgh Consolidation Coal Co.
Presented by: Clayton G. Ball, Paul Weir Co.

FRIDAY, OCTOBER 30

Chairman: W. S. Major, Dravo Corp.

10:00 am
The Midwest Solid Fuels Situation: Walter H. Voskuil, mineral economist, Illinois State Geological Survey.

11:00 am
Natural Gas Supply for Chicago and the Midwest: K. B. Nagler, vice president, operation, Peoples Gas Light & Coke Co.

12:30 pm

Luncheon Meeting

Presiding—L. S. Wescoat, president, Pure Oil Co.

Research in Fuels for Internal-Combustion Engines: Robert E. Wilson, chairman, Standard Oil Co. of Ind.

Afternoon Meeting

Chairman: R. B. Engdahl, Battelle, Memorial Institute

2:00 pm
Coal Mining Developments: J. D. A. Morrow, president, Joy Mfg. Co.

3:00 pm
Coal Preparation Developments: Robert E. Sloan and W. McCulloch, Roberts & Schaefer Co.

EJC Survey Studies Engineer Salary Range, Employment, Education

Now available on request to the Bureau of Labor Statistics, Washington 25, D. C., is a 48-page mimeographed report on *Employment, Education, and Income of Engineers, 1949-50 — A Survey of Engineering Society Members of Full Professional Grade.* The survey was made by Engineers' Joint Council on behalf of the Dept. of Defense. It covers returns from 55,000 engineers who hold full professional grades of membership in national engineering societies, or more than half of society members of full professional grade. Of the returns, about 40 pct came from mechanical engineers, 25 pct from civils, 15 pct from electricals, 12 pct mining and metallurgical, and 7 pct chemical. The groups with the lowest median age, 42 years, were the chemical and mining and metallurgical engineers. For the mechanicals the median was 43 years, for the electricals 46, and for the civils 47. There were only 56 women in the 55,000 considered in the survey.

	35 to 39	40 to 44	45 to 49	50 to 54	55 to 59	60 to 64	65 to 69	70 Up
Ph.D.	$8800	$8500	$9300	$9400	—	—	—	—
M.S.	7000	8200	8500	9100	$10,100	$10,800	—	—
B.S.	6600	7600	8500	9200	10,300	11,000	$10,200	$8300
None	6000	6500	7700	9000	9400	8300	10,000	—

Of the total in all fields, 3.7 pct held doctor's degrees (4.3 for mining and metallurgical engineers); 16.4 pct held master's degrees (16.5 pct for mining and metallurgical); 59.5 pct held bachelor's degrees (61.9 mining and metallurgical); and 20.3 pct held no degree (17.3 pct for mining and metallurgical).

Median incomes of those reporting ranged as follows:

Self employed, employer, or owner of business, $14,700; self employed, independent consultant, $10,700; private industry, $8400; educational institutions, $7100; government, $6600.

Among mining and metallurgical engineers, 81.7 pct were in private industry; 6.2 pct worked for governmental agencies; 5.2 pct were educators; 3.7 pct were independent consultants; 1.8 pct were employers or owners; and 1.3 pct worked for foundations.

Median incomes of various kinds of engineers, 35 years of age and over, 44,000 engineers reporting: Chemical, $9000; mechanical, $8400; mining, metallurgical, electrical, $8000; civil, $7100.

Median annual income of mining and metallurgical engineers, by age groups and highest degree earned are listed in table shown below.

Many other data are given in the complete report.

Engineering Growth Theme of Gathering

Theme of the joint meeting October 14 to 17 of the American Society for Engineering Education and the Engineers' Council for Professional Development will be "Civilization is Dependent Upon the Growth of the Engineering Profession."

All engineers are being encouraged to participate in the program which is to be held at the Statler Hotel, New York City. Members and others are urged to write to the secretaries of their societies for full information on the program. An announcement will be ready for distribution around September 15.

Around the Sections

• **The Lima-Peru Section** celebrated one year of activation at its First Annual Meeting at the Lima Country Club with some 200 persons attending the affair. Following cocktails, a banquet was held in the main hall of the club. The 6 x 5 ft local section banner, made under supervision of the Woman's Auxiliary, was put on display for the first time. Carl W. Westphal, Chairman, reviewed the section's history. The section, which now totals 143 members, grew out of the joint idea of ten engineers. Also addressing the meeting were Ernesto A. Baertl, Henry F. Stubbs, and Mrs. Lawson P. Entwistle. Before the remainder of the evening was devoted to dancing, a champagne toast was drunk to the future of the section. Erwin Jones provided entertainment with magic that displayed professional skill.

Among those seated at the head table during the recent Annual Meeting of the Lima, Peru Section were: Mrs. Howard Nicholsen; Ernesto A. Baertl, Section Vice Chairman; Mrs. Lawsen P. Entwistle, Woman's Auxiliary Chairman; Carl W. Westphal, Section Chairman; Mrs. Cyril L. Fleishman; and Lawsen P. Entwistle. The Section's new banner hangs from the wall.

• Two films of more than passing interest to mining men are available. The first, produced by Heyl & Patterson Inc., Contracting Engineers, 55 Water St., Pittsburgh, Pa., is called *Heavy Bulk Materials Handling.* **It's a 16 mm sound film in full color and shows the various pieces of equipment used in handling coal and ore from mine to final use. It is not a brief for Heyl & Patterson and is objective in presentation.**

• The second film is put out by Dow Chemical Co., Public Relations Department, Midland, Mich., and is called *Treat Wood Right.* **The movie is 16 mm; in color and sound. It attempts to make users of wood aware of the advantages gained by correct employment of wood resources. It advances the idea of treated wood for the control of insects and decay, and runs for approximately 20 minutes.**

Personals

FRED H. BUNGE

Fred H. Bunge has left the Colorado School of Mines Research Foundation Inc. and is with the research laboratory of The M. A. Hanna Co., Hibbing, Minn.

Gerald Letendre has been elected a vice president of the Canadian Institute of Mining and Metallurgy.

Charles S. Phillips, formerly mine superintendent with Virginia-Carolina Chemical Corp., Nichols, Fla., is now with Kaiser Bauxite Co., Spur Tree, Jamaica, B.W.I.

George P. Swift, AIME director from 1949 to 1952, became national president in July of the American Electroplaters Society. **Ralph A. Schaefer** was elected vice president.

J. B. Johnson has been elected a vice president and a member of the executive committee of Hercules Powder Co., Wilmington, Del. Succeeding Mr. Johnson as general manager is **John M. Martin. LeRoy Keane,** director of sales, and **Harry V. Chase,** director of operations, have both been appointed assistant general managers. **Jack D. Hayes, Jr.,** has been made director of operations and **Clarence W. Ballard,** director of sales.

Warren E. Wilson, president of South Dakota School of Mines and Technology, has been appointed director of the Engineering Sciences Division of the Army Office of Ordnance Research at Duke University.

Andre L. Brichant, having accepted appointment by the United Nations Technical Assistance Administration, is in Indonesia as mineral resources expert.

William J. McCaughey, Professor Emeritus of mineralogy at Ohio State University, was awarded the honorary degree of Doctor of Science at Ohio State University's 1953 graduation. He received this honor in recognition of his work since 1911 in training thousands of students in mineralogy and physical chemistry.

Merle H. Guise has returned to New York from Brazil where he has been exploring for tin, tantalite, columbite, and other metals. He headed one expedition up the Araguari River into the interior of the Territory of Amapa.

Hugh R. Van Wagenen is consultant to the mining division of Jet-Lube Inc., Los Angeles.

Frank A. Downes has retired as vice president of The Dorr Company, but continues as a director.

Robert D. O'Brien, former manager of Highland-Surprise Consolidated Mining Co., Kellogg, Idaho, is in Japan. He is assistant territorial supervisor of export sales for The Eimco Corp.

Alfred R. Flinn, assistant superintendent of The New Jersey Zinc Co.'s Sterling Mine, Ogdensburg, N. J., has been transferred to the New York office as resident mining engineer.

ELLIOTT J. ROBERTS

Elliott J. Roberts, for 12 years director of research of The Dorr Company, has been promoted to the newly created position of technical director in charge of research, development, and the Westport Mill. Mr. Robert's former position is now held by **Bryant Fitch.**

Ralph D. Parker has been elected president of Canadian Nickel Co. Ltd., subsidiary of International Nickel Co. of Canada Ltd.

Paul Linz has become chairman of the board of Mercantile Metal & Ore Corp., New York.

George B. Clark has been advanced to professor at the department of mining and metallurgy at the University of Illinois.

John H. Lucas, after a leave of absence for a year from Kennecott Copper Corp., Ray, Ariz., has returned and been made chief of party for pit engineering. He has been studying at the University of Arizona where he obtained a degree in mining engineering and in mining geology.

Reserve Mining Co.'s huge taconite processing plant, under construction near Beaver Bay, Minn., has been named the E. W. Davis Works in honor of **Prof. E. W. Davis,** director of Mines Experiment Station of the University of Minnesota. With the University since 1912, he is recognized as a leading authority on taconite and has played a key role in the development of many beneficiation processes.

C. M. Donahue, manager of the mining department and international div., and **E. G. Sanner,** manager of manufacturing, at Mine Safety Appliances Co., Pittsburgh, have been elected vice presidents.

D. Moreno and **E. H. Monroe** are in charge of the recently established New York City mining and metallurgical div. of Brown & Root Inc.

C. B. Hunt succeeds **Robert W. Webb** who retired August 31 as executive director of The American Geological Institute which operates under the National Research Council in Washington, D. C. The AIME is one of the eleven member organizations of AGI.

GALE A. HANSEN

Gale A. Hansen has finished his contract with Cia. Minera Aguilar, Prov. de Jujuy, Argentina, where he has been mine superintendent for the past three years. His address is Box 4, Heber City, Utah.

Obituaries

Kleber Bordez (Member 1949) died. Mr. Bordez was born in France and studied at the Ecole Nationale Superieure des Mines de St. Etienne. Seven years he was director of a flotation plant in Turkey and for ten years director of the Mines d'Aouli in French Morocco. Since 1945 he had been director general of the Cie Francaise des Mines du Laurium in Greece.

Harlan S. Emlaw (Member 1898) died on Feb. 5, 1953. He was a retired mining engineer, formerly president of American Potash & Chemical Corp. Born in Grand Haven, Mich., Mr. Emlaw received his E.M. from Michigan College of Mining and Technology in 1895. He first worked as a surveyor in Cripple Creek, Colo.; then as an engineer with Basin & Bay State Mining Co. in Montana and with the Washoe plant of Anaconda Copper Mining Co. For several years Mr. Emlaw was general superintendent of mines for Cerro de Pasco Corp. in Peru. At various times he was vice-president of the Fresnello Co. and president of Golden Queen Mining Co., Tri-State Zinc Inc., and Buell Engineering Co. Mr. Emlaw was a Legion of Honor Member of AIME.

Frederick Garrison Lasier (Member 1907) died Jan. 29, 1953. Mr. Lasier was a consulting engineer and mine owner. He was born in 1878 in Detroit and received his B.S. and E.M. from Michigan College of Mines. After studying at Columbia University for a short period, Mr. Lasier worked as an engineer with Cliff Mine, U. S. Steel Co., Iron Mountain, Mich., and then for various companies in Canada and Mexico. During World War I he was a captain with the U. S. Army Engineers. He was owner of the Clyde Gold Mfg. Co. and the Cuff Iron Mining Co. Mr. Lasier was a member of the Engineering Society of Detroit, the S. A. M. E., and the New York Engineers' Club. For the past several years he had made his home in Los Gatos, Calif.

Erskine Ramsay (Member 1888) died Aug. 15, 1953. Mr. Ramsay was a mining engineer, industrialist, and philanthropist. Born in 1864 in Six Mile Ferry, Pa., he graduated from St. Vincent's College, Pa., in 1883. Trained under his father, who was a coal mine superintendent, Mr. Ramsay as a youth worked for as little as $2.00 a month, yet he became a millionaire while in his early forties. At the age of 19, he became superintendent of the H. C. Frick Coke Co.'s Monastery Mines; the next year he was made superintendent of the Morewood Coke Co. and the South West Coal and Coke Co. In 1887 Mr. Ramsay went to Birmingham as superintendent of the Pratt mines for the Tennessee Coal, Iron and Railroad Co. He became vice president and chief engineer in 1901 for the Pratt Consolidated Coal Co. which is now part of Alabama By-Product Corp. Later severing his association with coal mining, Mr. Ramsay entered into partnership with G. B. McCormack and became president of the Ramsay-McCormack Land Co. Mr. Ramsay held directorships in numerous Alabama industries and organizations. During World War I he was a Dollar-a-year man and a member of the Peabody Committee. He was president of the Alabama Board of Education from 1922 to 1941. His gifts to education in Alabama amounted to hundreds of thousands of dollars. Mr. Ramsay's best known invention, the shaking screen process, was never patented, but he patented 40 other devices used in coal mining. Although he received several honorary degrees and many other honors, the one Mr. Ramsay cherished the most was the William Lawrence Saunders Gold Medal of the AIME awarded to him "for bituminous coal mining inventions; for improvement in coke making that resulted in the establishment of the steel industry in Alabama; for administering large enterprises and for benefactions to educational institutions." Mr. Ramsay was a Legion of Honor Member of AIME.

Robert Holden Stewart (Member 1905) died Dec. 23, 1952 in Vancouver. He was called the "Dean of the profession of mining engineering" in British Columbia. "Pat" Stewart was born in St. John, New Brunswick, in 1874. His early education was received in England, at Blackheath and Cambridge. Returning to Canada, Mr. Stewart received his B.S. degree from McGill University in 1896. Soon afterwards he went to British Columbia where he worked as a surveyor for Le Roi Mining Co. in Rossland. Later, joining the staff of what was to become Consolidated Mining & Smelting Co. Ltd. of Canada, he made an examination of the Sullivan mine at Kimberly and it was on his recommendation that this property, now one of the greatest lead-zinc-silver mines in the world, was acquired. In association with others at Cominco, he was responsible for introducing the electrolytic zinc treatment method at Trail. In 1911 Mr. Stewart became general manager of Cominco and held this position until 1918 when he went into private practice as a consulting engineer. Among other companies, he was retained by Cominco, Texas Gulf Sulphur, and Canadian Exploration Ltd. Mr. Stewart was a member of the Canadian Institute of Mining and Metallurgy and served at different times as councillor, vice president, and president.

NECROLOGY

Date Elected	Name	Date of Death
1905	Julian Boyd	Mar. 8, 1953
1951	Grenville H. Grimwod	Unknown
1900	Donald F. Irvin	July 1, 1953
1921	Marvin Lee	July 7, 1953
1942	K. R. Paykull	June 18, 1953
1949	Kimball M. Williams	Unknown

Proposed for Membership

MINING BRANCH, AIME

Total AIME membership on June 30, 1953 was 19,010; in addition 1543 Student Associates were enrolled.

ADMISSIONS COMMITTEE

O. B. J. Fraser, Chairman; Philip D. Wilson, Vice-Chairman; F. A. Ayer, A. C. Brinker, R. H. Dickson, Max Gensamer, Ivan A. Given, Fred W. Hanson, T. D. Jones, George N. Lutjen, E. A. Prentis, Sidney Rolle, John T. Sherman, Frank T. Sisco, R. L. Ziegfeld.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

Alabama
Birmingham—Dahlen, Emile O. (R. C/S—J-M)
Arizona
Jerome—Kintzinger, Paul R. (R. C/S—S-J)
Phoenix—Strong, James E. (A)
Superior—Mannheimer, Louis H. (M)
California
Menlo Park—Greef, Edward B. (J)
Pasadena—Von Migula, Gerhard (M)
Riverside—Lenhart, Walter B. (M)
San Rafael—Mitchell, Miles J. (M)
So. Pasadena—Hughes, Herbert V. (M)

Colorado
Grand Junction—Sharp, John V. A. (J)
Florida
Plant City—Martinez, Federico (J)
Idaho
Soda Springs—Whiteside, Joseph L. (M)
Illinois
Alton—Crum, Eugene H. (A)
Wheaton—Gilbert, Bruce W. (R. C/S—S-J)
Indiana
Bloomington—Patton, John B. (M)
Kansas
Lawrence—Buchholtz, Herbert F. (R. C/S—S-J)
Maryland
Frostburg—Hartig, Philip, Jr. (M)
Massachusetts
Fall River—Hadfield, Jonathan P., Jr. (J)
Salem—Wall, Ernest F. (R. C/S—S-M)
Stoneham—Thomas, Peter N. (J)
Winchester—Kaufman, David (M)
Michigan
Iron River—Seymour, Oliver G. (R. C/S—J-M)
Ishpeming—Campbell, Dugald K. (R. C/S—A-M)
Ishpeming—Lindroos, Emert W. (M)
Minnesota
Hibbing—Boentje, John D., Jr. (R. C/S—J-M)
Hibbing—Young, Ernest A. (M)
Leoneth—Flaaten, Donald S. (A)
Missouri
Bonne Terre—Koen, Harold L. (M)
Joplin—Gaines, Richard V. (C/S—A-M)
Montana
Butte—Mattson, George A. (R. C/S—S-J)
Nye—Lukens, John L. (A)
Nevada
Overton—Morledge, Frederick L. (M)
New Jersey
Somerville—Wright, Donald H. (M)
New Mexico
Albuquerque—Severinghaus, Nelson, Jr. (R. C/S—S-J)
New York
Larchmont—Andersen, Axel B. (M)
New York—Schoen, Robert (J)
North Creek—Barton, Henry H. (M)
North Carolina
Burnsville—Simpson, Wendell H. (M)
Kings Mountain—Dill, William E., Jr. (J)
North Dakota
Grand Forks—Achttien, Donald B. (C/S—S-J)
Ohio
Cleveland—Anderson, Robert F. (R. C/S—S-A)
Columbus—Hanson, Glen E. (R. C/S—S-J)
Columbus—Riley, William E., Jr. (J)
Youngstown—Jones, Wilbur K. (A)
Oklahoma
McAlester—Dinsmore, Alexander M. (M)
Miami—Templain, Curtis J. (J)
Oregon
Albany—Brown, Lawrence L. (J)
Portland—Anderson, John Russell (R. C/S—A-M)
Pennsylvania
Pittsburgh—Karniski, Walter J. (A)
Pittsburgh—Ketchka, Edward R. (J)
Pittsburgh—Lewis, Walter H. (A)
Pittsburgh—Ruffner, John M. (R. C/S—S-J)
St. Clair—Thomas, Edward T. (M)
Tennessee
Mascot—Cameron, Sidney M. (J)
Morristown—Callaway, Harry M. (M)
Texas
Houston—Schoenike, Howard G. (C/S—A-M)
Utah
Cedar City—Horen, Arthur (C/S—J-M)
Lark—Hart, Owen J. (A)
Murray—Last, Arthur W. (R. C/S—S-M)
Ogden—Thurber, Horace K., Jr. (R. C/S—S-A)
Orem—Hansen, Don A. (J)
Salt Lake City—Hulse, Edward H. (A)
Salt Lake City—Pack, Douglas H. (R. C/S—S-M)
Virginia
Alexandria—East, William F. (A)
Washington
Spokane—Parsons, Lewis F. (A)
Africa
Southern Rhodesia, Gatooma—Lancaster, John G. (J)
Tunisia, Tunis—Savin, Robert (M)
Asia
Saudi Arabia, Jedda—Grala, Peter (A)
Saudi Arabia, Jedda—Lloyd-Jones, Clopton F. (M)
Canada
Ontario, Pamour—Cox, Bart (J)
Ontario, Toronto—Ward, Stanley H. (M)
Europe
England, London—Robinson, Alan J. (J)
Finland, Imatrankoski—Snellman, Matts G. (C/S—S-A)
Mexico
San Luis Potosi—Mathis, Gerald C. (M)
South America
Argentina, Buenos Aires—Tolderlund, Otto (M)
Chile, Copiapo—Cossoro, Carlos F. (R. M)
Peru, Atacocha, Cerro de Pasco—Quintana S., Jorge (J)
Peru, Cerro de Pasco—Ward, Hector J. (M)

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Coming Events

Sept. 13-16, Electrochemical Chemical Society Inc., Ocean Terrace Hotel, Wrightsville Beach, N. C.

Sept. 18-19, AIME, Southeast Section, Asheville, N. C. (All Sectional).

Sept. 21, ASME, Industrial Instruments and Regulators Div., and Instrument Society of America, exhibit and joint conference, Sherman Hotel, Chicago.

Sept. 21-24, American Mining Congress Mineral Mining Convention, Olympic Hotel, Seattle.

Oct. 1-2, AIME, Regional Meeting, Pacific Petroleum Chapter, Ambassador Hotel, Los Angeles.

Oct. 6, AIME, National Open Hearth Steel Committee, Buffalo Section.

Oct. 7, AIME, National Open Hearth Steel Committee, Chicago Section, Phil Smidt's.

Oct. 7-9, National Assn. of Consulting Engineers, south central region, Mayo Hotel, Tulsa, Okla.

Oct. 8-9, Ninth National Conference on Industrial Hydraulics, Hotel Sheraton, Chicago.

Oct. 9, AIME, St. Louis Local Section, St. Louis University, St. Louis.

Oct. 9, AIME, National Open Hearth Steel Committee, Eastern Section, Fall Meeting, Warwick Hotel, Philadelphia.

Oct. 15-17, AIME, Second Annual Clay Minerals Meeting, University of Missouri, Columbia, Mo.

Oct. 15-17, AIME National Open Hearth Steel Committee, Southwestern Section, Fall Meeting, Baker Hotel, Dallas.

Oct. 18-21, AIME, Petroleum Branch Fall Meeting, Baker and Adolphus Hotels, Dallas.

Oct. 19-21, AIME, Institute of Metals Division, Fall Meeting, Hotel Allerton, Cleveland.

Oct. 19-23, Forty-first National Safety Congress and Exposition. Sessions on industrial safety, Conrad Hilton, Congress, Morrison, and Hamilton Hotels, Chicago.

Oct. 22, AIME, St. Louis Local Section, Engineers Club of St. Louis and ASCE, Engineers Club, St. Louis.

Oct. 22-24, The American Ceramic Society Inc. Sixth Pacific Coast Regional Meeting, Palace Hotel, San Francisco.

Oct. 27, Assn. of Consulting Chemists & Chemical Engineers, 25th Anniversary Annual Meeting, Hotel Belmont Plaza, New York.

Oct. 28-Nov. 2, AIME, El Paso Meeting, in cooperation with International Mining Days, Hotel Paso Del Norte, El Paso.

Oct. 29-30, AIME, ASME Fuels Conference, Conrad Hilton Hotel, Chicago.

Oct. 29-31, Annual Meeting of National Council of State Board of Engineering Examiners, Plaza Hotel, San Antonio.

Oct. 30-31, AIME, National Open Hearth Steel Committee, Deshler-Wallick, Columbus.

Oct. 30-31, AIME, Southeast Section, Knoxville, Tenn. (All Sectional).

Nov. 6, AIME, National Open Hearth Steel Committee, Pittsburgh Section, William Penn Hotel, Pittsburgh.

Nov. 6-7, Annual Fall Meeting of the Central Appalachian Section, AIME. To be held jointly with the West Virginia Coal Mining Institute at The Greenbrier, White Sulphur Springs, W. Va.

Nov. 13, AIME, St. Louis Section, Coal Meeting, York Hotel, St. Louis.

Nov. 29-Dec. 4, ASME, Annual Meeting, Statler Hotel, New York.

Dec. 2-4, Electric Furnace Steel Conference, Netherland Plaza Hotel, Cincinnati.

Dec. 11, AIME, St. Louis Section, York Hotel, St. Louis.

Dec. 13-16, American Institute of Chemical Engineers, Annual Meeting, Hotel Jefferson, St. Louis.

Dec. 28-29, Annual Chemical Engineering Symposium, University of Michigan, Ann Arbor.

Feb. 15-18, 1954, AIME, Annual Meeting, Mining and Petroleum Branches, Hotel Statler; Metals Branch, Hotel McAlpin, New York.

Advertisers' Index

*** Previous Issues**

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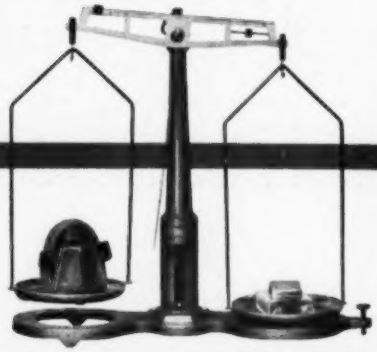

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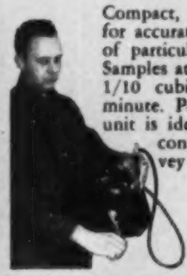

Compact, easy to use for accurate sampling of particulate matter. Samples at the rate of 1/10 cubic feet per minute. Portable, the unit is ideal for dust control and survey work in the mine.

M.S.A. DUST-VUE MICROPROJECTOR

Show collected dust particles magnified to 1000 times on ruled, translucent screen. Permits accurate, rapid counting and size determination. Relieves eye strain.